

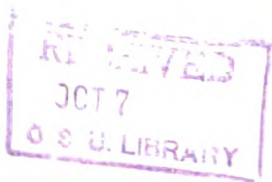
DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES

JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

MINING AND MILLING TUNGSTEN ORES



BY

WM. O. VANDERBURG

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DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

MINING AND MILLING TUNGSTEN ORES^{1/}

By Wm. O. Vandenburg^{2/}

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^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6852."

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INTRODUCTION

This report on the mining and milling of tungsten ores is the second part of a general study of the domestic tungsten industry. The first part, comprising general information on tungsten, has been printed as Bureau of Mines Information Circular 6821.

The purpose of these papers is to increase efficiency in the mining, preparation, and utilization of the tungsten resources of the United States. The tungsten industry forms so small a part of the vast mining and metallurgical operations of the Nation that private metallurgists have not had enough inducement to give special attention to the recovery of tungsten minerals from the ores, and many of the small companies are financially unable to undertake research work. A study of the milling of tungsten ores reveals that further improvement is possible, especially in the treatment of slimes. Recent experimental work by the Bureau shows that the flotation process is applicable to certain tungsten ores.

The future of the tungsten industry in the United States depends to a great extent on utilization of the low-grade deposits of the contact-metamorphic type.

ACKNOWLEDGMENTS

Thanks are due the owners and operators of the various companies engaged in the mining and milling of tungsten ores in the United States who without exception cooperated whole-heartedly in placing private data at the author's disposal and in discussing their problems.

TYPES OF TUNGSTEN DEPOSITS

Ore deposits are usually classified according to the processes involved in their origin. Tungsten deposits are represented in almost every type of deposit. These include: (1) Segregation deposits, (2) pegmatites, (3) replacement deposits, (4) contact-metamorphic deposits, (5) veins, and (6) placers.

Segregation Deposits

As the name implies, this type of deposit is formed by the segregation of certain minerals from an igneous mass on cooling. The minerals thus differentiated are present in greater amounts in the deposit than in the rock as a whole.

Segregation deposits are characterized by their irregular form and their freedom from fissures. These deposits are a part of igneous intrusions, and the gangue minerals are those of the parent rock. A tungsten deposit of this

type occurs in the Whetstone Mountains, Cochise County, Ariz., where small, flat crystals of wolframite are distributed through granite in irregular masses that lie close to small, irregular gash veins.^{3/}

Pegmatites

Pegmatite deposits are related closely to segregation deposits, as they are also end products of crystallization that have been thrust into openings in rocks already consolidated. Pegmatites are commonly found in dikes but sometimes show pipelike or other irregular forms. The constituent crystals of pegmatites generally are larger than the crystals of igneous rocks or veins; quartz, feldspars, and mica predominate. In addition to these minerals, which are also present in the parent granite from which the pegmatites are derived, pegmatites may include a great variety of other minerals, such as tourmaline, apatite, spodumene, amblygonite, garnet, beryl, columbite, triphylite, lithiophilite, cassiterite, and wolframite.^{4/} These latter minerals if present in the parent rock occur in very small amounts, so that pegmatites may be considered a type of segregation deposit. On the other hand, some pegmatites which consist largely of quartz show the characteristics of veins.

The best examples of tungsten-bearing pegmatites in the United States are in the Harney Peak and Tinton districts, South Dakota. Although these pegmatites are important as a source of some rare minerals and gems, they have been of minor importance as tungsten producers.

Replacement Deposits

Replacement deposits are formed where solutions have dissolved the original rock and material of different composition is deposited in the space left by the solution. Deposition may follow solution so closely that the form and texture of the original material are preserved. Because of their solubility, carbonate rocks are favorable for replacement.

The best-known examples of this type of tungsten deposit in the United States occur in the northern Black Hills near Lead, S. Dak., where flat-lying dolomitic limestone has been partly replaced by silica and wolframite. The ore bodies average about 2 feet in thickness, although they range from lenticular seams a few inches thick to large lens-shaped bodies about 6 feet thick.^{5/} In general, replacement deposits of tungsten ores are irregular in form and erratic in mineral distribution.

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- ^{3/} Hess, Frank L., Note on a Wolframite Deposit in the Whetstone Mountains, Ariz.: U.S.Geol. Survey Bull. 380, 1909, pp. 164-165.
- ^{4/} Connolly, J. P., and O'Harra, C. C., The Mineral Wealth of the Black Hills; South Dakota Sch. Mines Bull. 16, 1929, pp. 220-221.
- ^{5/} Runner, J. J., and Hartmann, M. L., The Occurrence, Chemistry, Metallurgy, and Uses of Tungsten with Special Reference to the Black Hills of South Dakota; South Dakota Sch. Mines Bull. 12, 1918, pp. 67-68.

Contact-Metamorphic Deposits

Contact-metamorphic deposits are formed in intruded rocks of calcareous composition by solutions and gases emanating from the intruded magma. Metamorphism destroys the original composition of the carbonate rocks; during the process of destruction an aggregate of characteristic contact-metamorphic minerals is developed--garnet, epidote, calcite, diopside, actinolite, scheelite, and numerous silicates of calcium, magnesium, and iron. This type of tungsten deposit is readily recognized by these minerals. The tungsten mineral in contact-metamorphic deposits is invariably scheelite; it occurs as grains ranging from the size of a pinhead to 1 inch or more in diameter, but the smaller sizes usually predominate.

Mineralization of the contact-metamorphic deposits generally follows the structure of the original sedimentary beds. The scheelite usually is distributed throughout the beds more uniformly than tungsten ores in the other types of deposits.

Nearly all the known contact-metamorphic tungsten deposits in the United States were discovered during the World War when the demand for tungsten was great and the prices rose to unprecedented heights. In recent years most of the domestic production of tungsten concentrates has been derived from the contact-metamorphic deposits of the Great Basin region. Contact-metamorphic deposits have been worked in the Mill City and Nightingale districts, Nevada, and near Bishop, Calif.

Veins

Veins are formed by solutions that have filled cracks or fissures within the rocks. The mineralizing solutions in some places have replaced the wall rocks, but ordinarily such replacement is not so extensive as to obliterate the relationship of these deposits to the fissures. In other places the walls of the veins are frozen to the vein matter. Veins containing tungsten ores may occur in single crevices or in two or more parallel zones of shearing. If shearing has been intense a series of interlacing veins or stockworks may be formed. Tungsten veins are subject to all the vagaries common to veins containing silver, gold, or other ores, such as branching, alternate pinching and swelling, brecciation, and sudden changes of direction. The tungsten minerals may be any of the important ones, with wolframite a little more common than ferberite, hubnerite, or scheelite. Little byproduct tungsten is produced in the United States from veins carrying ores of other metals, although gold, usually in small amounts, is found associated with scheelite and silver in the Wolframite group tungsten minerals. In veins in other parts of the world gold, molybdenum, bismuth, tin, and copper occur with tungsten ores. In general, tungsten ores in veins are erratic in form, size, and distribution of tungsten content.

Quartz and calcite are the most abundant gangue minerals in this type of deposit, although virtually all minerals ordinarily associated with veins may be present. Inclusions of wall rock are often found in the vein filling.

Representative vein-type tungsten deposits occur at Boulder, Colo., at Atolia, Calif., and in the Silver Dyke district near Mina, Nev. In the past the major production of tungsten concentrates in the United States has been derived from veins.

Placers

Placer deposits are formed by the agents of erosion - heat and cold, impact of sliding and water-carried rocks, vegetable growth, and chemical decomposition in conjunction with the mechanical processes of natural concentration. Under proper conditions virtually all types of tungsten deposits of sufficient size are possible sources of placer tungsten. As erosion takes place the tungsten minerals, like gold, platinum, cinnabar, and tin minerals, lodge at the foot of slopes or in streams along with sand and gravel. The minerals associated with tungsten placers are similar in that they are not readily soluble in water or natural solutions. Tin is commonly associated with tungsten minerals in various parts of the world. Tungsten placers formed on slopes are likely to have the tungsten minerals uniformly distributed throughout the gravel, whereas those formed by swift streams are likely to be irregular and their minerals may lie in pockets.

Not all tungsten deposits in place are associated with placers. Since tungsten minerals are extremely brittle they may have been ground to an impalpable powder by intensive mechanical attrition and carried away in the drainage. Moreover, if conditions were favorable, the tungsten minerals may have been dissolved and thus carried away.

From the standpoint of world production of tungsten concentrates, placers have been the most important producers, the wolframite placers in the provinces of Kiangsi and Kwangtung, China, being the most productive. The best-known tungsten placers in the United States are in the Boulder district, Colorado, the Atolia district, California, and the Snake Range near Osceola, Nev. All these placers were worked intensively during the World War and have produced considerable tungsten concentrates.

PROSPECTING

Where tungsten deposits have been exposed to the agents of erosion the formation of placers would be expected. Conversely, where placer tungsten has been found the search for the source of the placer tungsten offers interesting prospecting possibilities. Since placer deposits are relatively easy to discover and since their products can be worked easily and concentrated with little equipment, placers are usually the first resources of a region to be exploited. Placer tungsten deposits have been traced to their sources by prospecting with a pan as one would prospect for other heavy minerals like gold or platinum.

An experienced prospector will trace the source of placer tungsten by panning the gravel at intervals, following stream channels, gulches, or other natural topographical drainage courses of a region. When the tungsten colors

cease to appear in the pan the immediate topography is scrutinized carefully for indications of a vein or other form of deposit. Outcrops of tungsten deposits are not always conspicuous; where there are no surface indications of a vein trenching to bedrock may disclose the source of the placer material. The prospector must not conclude, however, that where placer tungsten is found the lode from which it was derived must be present, since some of the original shallow deposits may have eroded away entirely. In the search for tungsten ores panning along exposed outcrops or below the probable location of debris-covered outcrops is the cheapest way of locating ore shoots and determining their length.

In searching for tungsten ore bodies in undeveloped districts the prospector has another indication of the possible or probable presence of an ore body, namely, a body of rock of the kind usually associated with tungsten ores. Tungsten deposits, like tin and molybdenum deposits, almost always are associated with siliceous igneous rocks, especially granites or acid pegmatites. In this respect they differ from many other minerals. The converse is not true, however, as large areas of granitic rocks all over the World occur unassociated with tungsten deposits. Most of the tungsten deposits in place lie wholly within or at the margins of the acid igneous rocks, although some lie within sedimentary rocks such as slates, shales, or schists. Where the tungsten deposits lie within the latter rocks the underlying granitic rock is not far away even though not in evidence in the immediate vicinity.

Geological maps and bulletins of both State and Federal geological surveys are very useful in locating areas of light-colored (siliceous) rocks where tungsten ores are apt to occur.

In the search for the contact-metamorphic type of tungsten deposits, the beds may be traced from the float minerals such as garnet. Knowledge of the common minerals associated with tungsten ores and ability to recognize them readily in the field are important aids in prospecting. The contact-metamorphic deposit at Pine Creek, Calif., was found as a result of the discovery of garnet on claims originally located for silver. That scheelite was associated with garnetiferous rocks became generally known to the western prospector during the World War when prospecting for tungsten was very active. This knowledge resulted in the discovery of many new deposits of the contact-metamorphic type.

The best clue, however, in prospecting for tungsten is the presence of one of the tungsten minerals. The wolframites are readily recognized by one who has studied the physical characteristics of specimens. Scheelite is harder to identify; it is similar in appearance to some common minerals like quartz and calcite, and its high specific gravity may be attributed to barite.

Tungsten minerals are not readily oxidized to secondary minerals, and outcrops of tungsten ores generally do not show prominent stains that indicate the presence of tungsten minerals.

Because of the occurrence of placer tungsten deposits, some geologists have argued that tungsten minerals are resistant to weathering, while others have contended that they are less resistant to weathering than is generally supposed.

Experiments^{6/} have shown that on the whole tungsten minerals are somewhat soluble and that under certain conditions they dissolve rather readily to form soluble salts or colloidal compounds and thus may be removed from outcrops. As Gannett's experiments show, carbonate solutions do not attack tungsten minerals, and any enrichment of tungsten deposits by such solutions is caused by the removal of valueless material, thereby increasing the grade of the ore near the surface.

Tungsten may be leached from its ore bodies without being redeposited below, and secondary enrichment is difficult to interpret since the secondary minerals are the same as the primary ones. During alteration the secondary minerals generally remain near the primary minerals. From a chemical standpoint tungsten minerals may be classed with the nonmigratory minerals.

Prospecting Equipment

At the Cold Springs mine, Nederland, Colo., diamond drills are used in prospecting underground. Instead of the regulation diamond bit a firthite bit is used. Firthite is an extremely hard compound consisting of tungsten carbide cemented with cobalt under high temperature. The firthite fragments are brazed by an acetylene welder into the end of Tobin bronze core barrels one eighth inch thick. Care must be exercised in setting the firthite, as excessive heating would destroy its effectiveness. A complete bit can be made for about \$75. In drilling fissured rock the firthite bit is more effective than the diamond bit, as the attendant danger of losing the settings is not such an important factor. Further experimental work under field conditions is necessary to determine the general adaptability of the firthite core drill.

In certain types of tungsten deposits the deep-hole hammer drill may be employed to advantage. For prospecting contact metamorphic deposits, particularly for determining extensions of known beds and the presence or absence of parallel beds, the hammer drill could be employed, in some cases at less cost, than other methods of prospecting. The applicability and limitations of prospecting by the hammer drill have been pointed out in a paper published by the Bureau of Mines.^{7/}

Geophysical methods of prospecting have not been applied to tungsten deposits so far as the author is aware. Since the contact-metamorphic tungsten deposits contain magnetic minerals associated with scheelite, investigation may demonstrate that some type of magnetometer is suitable for outlining such deposits. Resistivity methods of prospecting may also be of use in exploring the known contact-metamorphic tungsten deposits in the Great Basin region.

^{6/} Gannett, R. W., Experiments Relating to the Enrichment of Tungsten Ores: Econ. Geol., vol. 14, 1919, pp. 68-78.

^{7/} Knaebel, J. B., Sampling and Exploration by means of Hammer Drills: Information Circ. 6594, Bureau of Mines, 1932, 32 pp.

MINING METHODS

Round Valley Tungsten Co. Mine

The Round Valley tungsten mine is 11 miles west of Bishop, Calif. The climate and altitude are such that operations can be carried on throughout the year without difficulty. Electric power is obtained from the hydroelectric plants of the Southern Sierras Power Co. on Bishop Creek 7 miles from the mine. The average cost of power is 1 1/2 cents per kilowatt-hour.

The deposit is of the contact-metamorphic type, and the only economic mineral in the ore is scheelite. The country rock is largely granite with masses of limestone which have been mineralized by the intrusion of the granite. The limestone is altered to tactite,^{8/} and in the masses of tactite the scheelite is disseminated as small crystals. The tactite body is about 125 feet wide and 200 feet long. The contact of the tactite and granite is extremely irregular. The average grade of the ore mined is 0.5 percent WO₃. The structural and economic conditions favor the gloryhole method of mining, which is employed.

Since the tactite outcrops on the side of a hill no elaborate development work was required. The deposit has been developed by 3 levels, of which 2 are adits, driven 50 feet apart. The working level is connected to the surface by a single-compartment shaft 100 feet deep inclined 75°. The shaft is 7 by 6 feet in the clear and is timbered with 8- by 8-inch square-sets.

Mining is confined to the glory hole, which centers about seven mill holes; these are started by belling out the tops of raises and drawing off the broken ore at the level below.

The shaft crosscut and the branches therefrom to the transfer raises connecting with the glory hole are 5 by 7 feet in section and untimbered. The raises are 5 by 6 feet in section and also are untimbered.

Drilling is done with jackhammer machines using 7/8 inch, hollow, hexagonal steel. Drill steels are hand-sharpened, and the Carr bit is used. Drilling is done dry. The hole in each drill steel terminates 1 inch behind the cutting edge of the bit to blow the holes free from cuttings more quickly. For blasting, 40-percent-strength gelatine dynamite is employed with no. 8 caps attached to tape fuse. Compressed air for drilling is supplied by an 8- by 10-inch Sullivan compressor.

The large boulders of ore broken in the glory hole are block-holed. The mill holes are kept full of ore to prevent boulders from falling in and choking them; otherwise, the expense of freeing choked ore passes would increase

8/ The term "tactite" has been proposed by Hess to designate the products of contact metamorphism in this type of tungsten deposits.

the mining cost considerably. The ore that does not run into the mill holes is shoveled in by hand. The walls of the glory hole stand well without sloughing, so that dilution of the ore with waste rock is not important.

The ore is drawn from the mill-hole raises into cars through chutes equipped with plank gates sliding in wooden cleats. The cars have a capacity of 1 ton each, and they are equipped with roller bearings. Track is made of 12-pound rails laid to a gage of 18 inches. Tramping is done by hand, and the ore is dumped from the cars into a 50-ton ore pocket at the shaft. Over the ore pocket is a grizzly with 10-inch openings made of 40-pound rails set in slots cut into 8- by 8-inch timbers. To prevent boulders from wedging between the rails of the grizzly the rails are laid with the flat side uppermost. Boulders that will not pass through the grizzly are broken by hand sledging.

From the underground ore pocket the ore is drawn off into a skip having a capacity of 1,000 pounds. The ore pocket is equipped with a plank-gate chute for loading. The skip dumps directly into a bin ahead of a crusher.

Eleven men are required to mine 120 tons of ore per day, working only one shift. The costs of mining on a basis of 74 tons per day is shown in table 1. The distribution of labor, power, and supplies is shown in table 2.

TABLE 1. -Mining costs per ton of ore hoisted, Round Valley mine

(Tons of ore hoisted: 2,250. Period covered: 1 month, 1931)

	Labor	Super- vision	Steel sharp- ening	Power	Explo- sives	Supplies	Total
Breaking.....	\$0.200	\$0.133	\$0.066	\$0.016	\$0.083	\$0.010	\$0.508
Tramping.....	.133	- - -	- - -	- - -	- - -	- - -	.133
Hoisting.....	.100	- - -	- - -	.012	- - -	.010	.122
Total.....	.433	.133	.066	.028	.083	.020	.763

TABLE 2. -Mining costs in units of labor, power, and supplies

A. Labor (man-hour per ton):

Supervision.....	0.053
Breaking.....	.320
Tramping.....	.160
Hoisting.....	.213
Blacksmithing.....	.053
Total.....	.799
Ore mined per man-shift (tons).....	10.01

I.C. 6852

B. Power and supplies:

Explosives, pound per ton of ore	0.33
Power, kw.-hr. per ton of ore.....	1.9
Air compression.....	1.0
Hoisting.....	.2
Total, kw.-hr.....	1.9

C. Labor percent of total cost.....	32.8
Power and supplies.....	17.2

Atolia Mining Co., Tungsten Placer

The Atolia district is in the western part of San Bernardino County, Calif., about 40 miles east of Mohave. Placer and vein deposits of scheelite occur in the district, but only the Atolia Mining Co. placer operations will be described.

Placer scheelite occurs in alluvium composed largely of detrital material resulting from the disintegration of quartz monzonite, the principal rock in the district. The scheelite nuggets range from particles the size of sand to ones several inches in diameter. Boulders of nearly pure scheelite weighing several hundred pounds have been found in the placer area.

Placer operations are confined to an area about a mile long and one half mile wide. The depth of the alluvium ranges from several feet to 150. The maximum depth that can be worked economically by power shovel is about 30 feet owing to the barren strata between the upper and lower channels in which the scheelite is concentrated. An important feature in the recovery of the scheelite is the fact that the detrital material is loosely cemented and blasting is not necessary.

The alluvium is mined with a gasoline-driven, revolving-type shovel mounted on caterpillars. The dipper has a capacity of 1 ton and is equipped with manganese-steel teeth, to which stellite tips are welded. The power shovel loads the material into two trucks having capacities of 7 3/4 and 5 1/4 tons, respectively. The trucks haul the placer ore to a dumping platform about 1,000 feet from the shovel. The trucks are driven up an inclined runway where the ore is discharged onto a grizzly with 6-inch openings placed over the top of a bin. The oversize material is broken by hand sledges.

Four men are required per 8-hour shift to mine about 250 tons of material. The total cost of mining and trucking per ton is 10 cents.

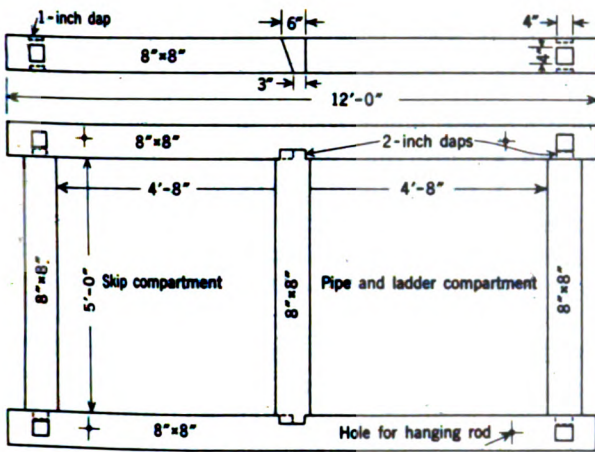


Figure 1.—Details of shaft timbering at Cold Springs mine.

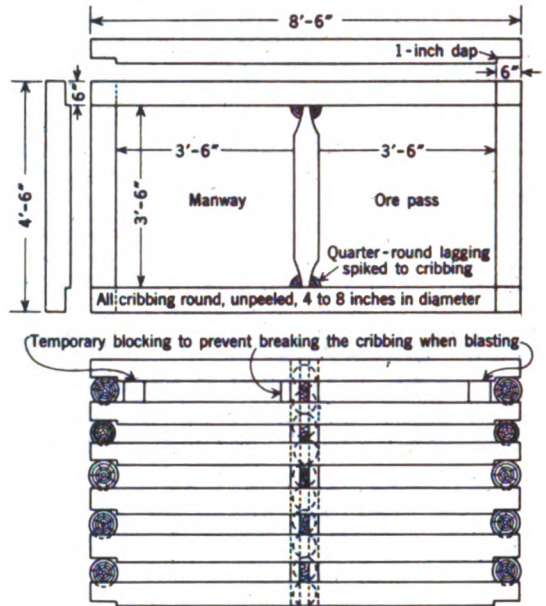


Figure 2.—Two-compartment raise showing timbering details at Cold Springs mine.

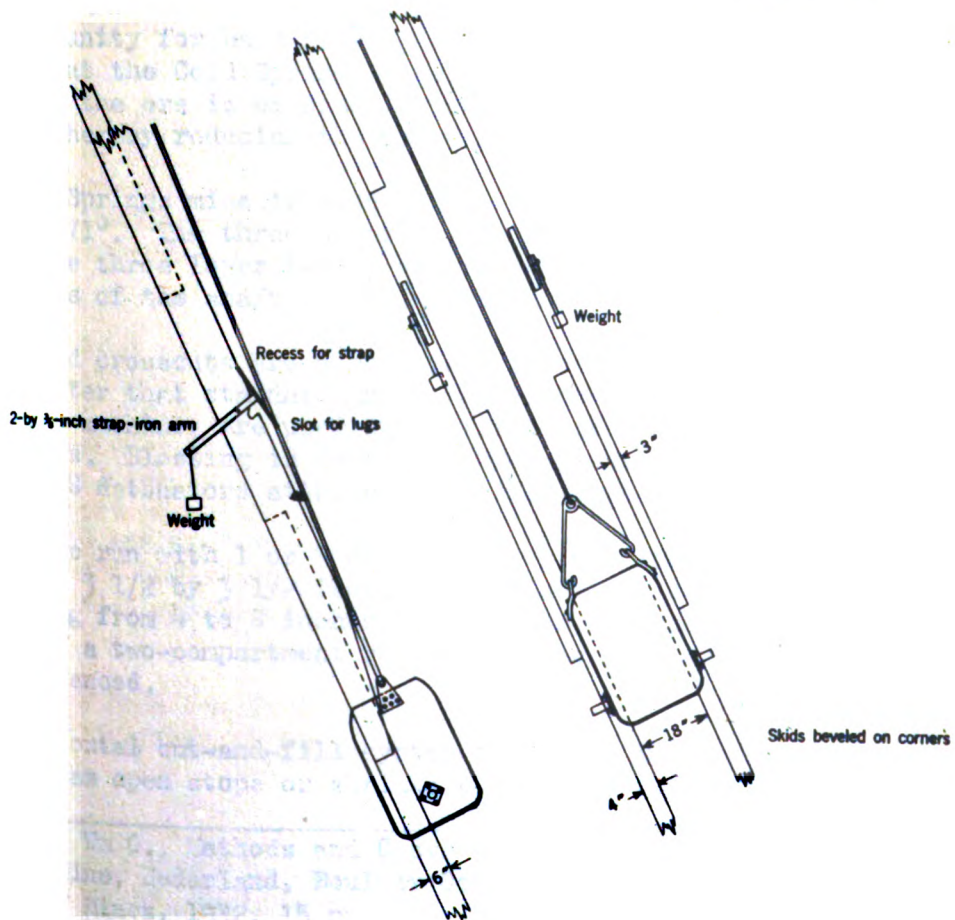


Figure 3.—Self-dumping bucket arrangement for sinking at Nevada-Massachusetts property.

Cold Springs Mine

The Cold Springs mine of the Wolf Tongue Mining Co. is 3 miles east of Nederland, Colo. The district is well-timbered, and round mining timbers are cut on company-owned land. Electric power for mining is purchased from the Public Service Co. of Colorado.²

At the Cold Springs mine the ferberite ores are mined by the cut-and-fill system. The ferberite occurs as lenses along vein fissures. The length of the lenses ranges from a few to 200 feet and averages about 80 feet. The width of the ore ranges from less than 1 inch to 6 feet; the average width mined is about 9 inches. The average dip of the veins is about 70°. The persistence of the lenses in width and mineralization is more dependable along the strike than the dip. The walls of the veins are granite traversed with a series of intersecting planes of weakness, so that the wall rock is likely to slough if openings are made for any distance over 6 feet along the dip.

The ferberite in the ore lenses is invariably of high grade, and the country rock is virtually barren of ferberite. These factors, in conjunction with the relative high value of the clean ferberite and the ease with which ore and waste are distinguished by visual inspection, afford an excellent opportunity for hand sorting and selective mining, both of which are practiced at the Cold Springs mine. The ores are fairly soft, so that if one side of the ore is exposed it can be removed easily with a moil and hand hammer, thereby reducing the waste mixed with the ore to the minimum.

The Cold Springs mine is opened by a two-compartment shaft 500 feet deep inclined 71°. The three upper levels have been driven at intervals of 50 feet and the three lower levels at intervals of 100 feet, inclined distance. Details of the shaft timbering are shown in figure 1.

Drifts and crosscuts are 5 by 7 feet in section. The ground varies so much in character that standard drill rounds cannot be employed. Column-mounted Leyner machines are used in drilling with 1-inch hexagonal steel with cross bits. Blasting is done with 40-percent-strength gelatin dynamite and no. 8 detonators attached to triple-tape fuze.

Raises are run with 1 or 2 compartments; the size of a compartment in either case is 3 1/2 by 3 1/2 feet. All raises are built of round pine timbers ranging from 4 to 8 inches in diameter. Figure 2 shows the timbering details of a two-compartment raise. Raises are carried up as the stops are advanced.

The horizontal cut-and-fill system of mining is preferred to other methods, such as open stop or shrinkage, for the following reasons:

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- 2 Vanderburg, Wm O., Methods and Costs of Mining Ferberite ore at the Cold Springs Mine, Nederland, Boulder County, Colo: Information Circ. 6673, Bureau of Mines, 1932, 15 pp.

(1) The ferberite occurs in narrow, rich veins, so that considerable waste must be broken to provide working space; (2) the walls are weak and require artificial support, which can be provided cheaply by utilizing the waste obtained by selective mining and hand-sorting waste from the ore in the stopes; (3) the higher-grade ferberite ores are at a premium so that it is desirable to obtain as high grade a product as is consistent with economy; (4) the ferberite can be recognized easily, and fine breaking is not required to separate ore from waste; (5) with adequate supervision the cut-and-fill system allows virtually complete extraction of the ore.

In mining no attempt is made to produce a large tonnage; complete extraction of the ore with minimum waste is emphasized rather than the tonnage broken, and in this respect the practice follows that ordinarily employed by lessees.

After an ore shoot has been opened by drifting a stope is started by taking a cut from the back of the drift to a height of 5 or 6 feet. In this work the ore and waste may either be broken separately or together. In the latter case the ore is hand-sorted. After a cut has been taken from the back the drift is studded with 3- to 12-inch round timbers placed on 5-foot centers between chutes and on 4-foot centers at the chutes and manways. Between the chutes the stalls are covered with a floor of round lagging 4 to 6 inches in diameter.

Access to stopes which average less than 100 feet in length is provided by a double-compartment raise at either end. Additional cribbed ore passes are built between the double-compartment raises at intervals ranging from 25 to 30 feet, center to center. Ore and waste are shoveled separately into the chutes by hand. The ore chutes are equipped with ordinary plank gates sliding in wooden cleats.

The stope is advanced by successive cuts 4 to 6 feet high carried along the full length of the ore shoot. Two methods of breaking are employed; where the ferberite occurs as a single band of rich ore stripping is employed, and where the ore and waste are intimately associated they are broken together on a sheet-iron shoveling platform and the waste is sorted from the ore by hand. In stripping the ore the waste on the hanging wall is carefully blasted first and allowed to drop into the fill. After the fill is leveled off by hand the exposed ore is broken onto a sheet-iron shoveling platform, either with small charges of explosives or with maul and hammer.

The amount of waste obtained by stripping and hand sorting is more than enough to provide the waste fill; the excess is shoveled into the chutes and transported to the surface.

The cribbed chutes and manways are built to a height of 3 or 4 feet above the level of the fill before each cut is taken along the back of the stope. The raise cribbing is not blocked against the solid rock but is held in place by large pieces of waste placed around the sides by hand. The waste thus placed also prevents the fine waste fill from entering the chutes.

Supplies are hoisted in the stopes by electrically driven tugger hoists.

Ore and waste are transported underground in end-dump plain-bearing steel cars holding about 1,200 pounds each. Track is made of 12-pound rails placed on a grade of 1/2 percent and a gage of 18 inches. At the shaft the cars are dumped over a pivoted pan chute directly into a skip which has a capacity of 2,400 pounds. Rails in the skipway compartment of the shaft are 4 by 6 timbers bolted lengthwise to the 3- by 8-inch footwall plates. Three-inch angle irons are bolted to the 4- by 6-inch timbers to provide track for the skip wheels.

At the surface the ore is dumped over an inclined bar grizzly with 1-inch openings. The fines pass into a bin and the oversize material onto a stationary sorting table. All pieces of ore are sorted from the waste by hand and thrown into the ore bin. Large pieces of ore and waste combined are hand-cobbed. The waste is shoveled into cars and hand trammed to the waste dump. Two sorters can sort and cobb about 20 tons of hoisted material per 8-hour shift.

The costs of mining during 1931 are shown in tables 3 and 4. The mining costs appear high, but it must be remembered that the ore mined is a concentrate produced by hand sorting. In addition the Cold Springs mine is a small-tonnage proposition, and the amount of development work and prospecting required is much greater than in average mining practice. Of the total material broken in stoping in 1931 about 70 percent was waste separated from the ore by stripping and hand sorting underground, 15 percent was waste rejected by hand sorting at the surface, and 15 percent was the ore sent to the mill.

TABLE 3. - Mining costs per ton of ore, Cold Springs mine

(Tons of ore hoisted: 1,125. Period covered: 8 months, 1931)

	Labor	Super- vision	Compressed air and steel	Power	Explo- sives	Timber	Other supplies	Total
Development.....	\$2.204	\$1.008	\$0.933	\$0.738	\$0.483	\$0.651	\$0.252	\$6.269
Mining.....	1.636	.672	.622	.494	.323	.432	.170	4.349
Tramming.....	.356	-	-	.616	-	-	.030	1.002
General expense.	3.840	-	-	-	-	.089	.513	4.442
Compensation insurance.....	1.110	-	-	-	-	-	-	1.110
Total.....	9.146	1.680	1.555	1.848	.806	1.172	.965	17.172

TABLE 4. - Mining costs in units of labor, power, and supplies, Cold Springs mine

A. Labor (man-hours per ton):

	<u>Development</u>	<u>Mining</u>	<u>Total</u>
Ore sorting.....	---	3.84	3.84
Hoisting.....	---	1.92	1.92
Blacksmithing.....	1.15	.77	1.92
Tramming.....	---	.71	.71
Shoveling.....	.93	.61	1.54
Breaking.....	1.83	1.24	3.07
General.....	1.83	1.24	3.07
Supervision.....	1.15	.77	1.92
Total man-hours per ton.....			17.99
Average ore mined per man-shift, (tons).....			0.445

B. Power and supplies:

Explosives, pounds per ton.....	4.48	Linear feet per ton
Timber: Cribbing 4 to 8 inches diam.....		26.7
Lagging, 3 to 4 inches diam.....		12.3
Stulls, 8 to 12 inches diam.....		3.3
Sawed stock, bd. ft. per ton.....		4.4
Power: Air compression.....		Kw. hr. per ton
Hoisting.....		29.3
Pumping.....		12.3
Ventilation and lighting.....		18.5
Total.....		1.5
		61.6

Percent of total
direct cost

C. Labor.....	63
Power and supplies.....	37

Silver Dyke Mine

The Silver Dyke tungsten mine is on the east flank of the Excelsior Mountains 12 miles southeast of Mina, Mineral County, Nev. The property is accessible by automobile road from Mina; all supplies are hauled from Mina by truck. Electric power for mining is purchased from the Mineral County Light & Power Co. at an average cost of about 3 cents per kilowatt-hour.

About 50 tons of scheelite ore are mined per day by the shrinkage-stope method. The Silver Dyke vein ranges from 30 to 140 feet in width and is traceable for about 4 miles on the surface. Within the vein the scheelite occurs in irregular lenses associated with quartz. The average width of the ore lenses is 3 feet and the maximum width 5 feet. The width of the vein to be mined is determined more by the scheelite content than by the structural characteristics. An effort is made to maintain the grade of the ore broken at 1 percent scheelite. Close supervision and numerous pannings have been found necessary to prevent the admixture of too much waste with the ore in mining operations. The rock stands well with only an occasional stull for support, so shrinkage stoping can be employed to advantage.

The mine is opened by a crosscut adit 871 feet in length. Drifts are driven 5 by 7 feet in section and generally are untimbered. Leyner machines using 1-inch hexagonal steel are used in drifting; 40 percent-strength gelatine dynamite is used for all blasting.

The length of the ore shoots ranges from 60 to 120 feet. Manways are carried up at both ends of an ore shoot to afford access to the stope. Chutes for drawing off the broken ore are spaced at 15-foot intervals. To provide support for the broken ore the floors of the stopes are timbered with stulls at 5-foot intervals; the stulls are overlain with plank lagging. Since the ore is mined by the shrinkage method the broken ore provides the working floor in advancing a stope. The broken ore is kept at a suitable distance from the back by drawing off the excess ore through plank chutes into cars. Hand-held stopers with 1-inch quarter-octagon steel are employed in stoping. Virtually all the ore broken in blasting is of such size that it will pass through a grizzly with 3-inch openings. The ore and waste are not hand-sorted at the Silver Dyke mine owing to their intimate association.

At the adit level the ore is drawn into end-dump cars through plank chutes equipped with plank gates. The cars are equipped with roller bearings, and each car has a capacity of 1 ton. A train of eight cars is hauled to the mine storage bin by a 110-volt storage-battery locomotive.

The direct costs for a 3-month period in 1931 are summarized in tables 5 and 6. Development work for the period consisted of 404 feet of drifting and 50 feet of raising. In addition, 1,140 tons of ore were mined, although approximately twice as much ore as this was broken but left in the stopes during the period and is not included in the tables.

TABLE 5. - Mining costs per ton of ore, Silver Dyke mine

(Tons of ore mined: 1,140. Period covered: 3 months, 1931)

	Labor	Super- vision	Compressed air, drills, and steel	Power	Explo- sives	Timber	Other supplies	Total
Development...	\$1.017	\$0.087	\$0.340	\$0.004	\$0.544	---	---	\$1.992
Mining.....	.636	.087	.268	---	.427	\$0.062	---	1.480
Transportation	.097	---	---	.015	---	---	---	.112
General Expense	.079	---	---	---	---	---	\$0.033	.112
Total	1.829	.174	.608	.019	.971	\$.062	.033	3.696

TABLE 6. - Mining costs in units of labor, power, and supplies, Silver Dyke mine

A. Labor (man-hours per ton):

	Development	Mining	Total
Breaking.....	0.758	0.660	1.418
Timbering.....	---	.360	.360
Shoveling.....	.881	---	.881
Hauling.....	.081	.060	.141
Supervision.....	.109	.109	.218
Miscellaneous.....	---	---	.510
Total.....			3.528
Average ore mined per man-shift (tons).....			2.27

B. Power and supplies:

Explosives, pounds per ton.....	3.59
Timber: Sawed stock, bd.-ft. per ton.....	1.76
Round timber, linear ft. per ton.....	.18
Power, kw.-hr. per ton.....	12.28
Air compression.....	11.54
Hauling.....	.61
Ventilation.....	.13
Total.....	12.28

Percent of total
cost

C. Labor.....	61.5
Power and supplies.....	38.5

Humboldt and Stank Mines

The tungsten mines of the Nevada-Massachusetts Co., Inc., are about 8 miles west of Mill City, Pershing County, Nev. This company has been the largest producer of tungsten concentrates in the United States for a number of years. The climate and topography offer no obstacle to mining operations. Electric power for mining is purchased from the Sierra Pacific Power Co. at a cost of about 2 cents per kilowatt-hour. The two mines operated by the company produce about 200 tons of scheelite ore per day averaging 1.1 percent scheelite.¹⁰

The formation in the Mill City district comprises a series of sedimentary rocks consisting of shales and sandstones, with intercalated beds of limestone and quartzite, which have been intruded by granite. Although the deposits are termed "contact-metamorphic" the ore occurs in tactite beds, not only at the contact of the granite but in some places 2,000 feet from the granite. Two of the limestone beds have been developed - the Sutton and the Stank. Of the two beds, the Stank has been the most productive; it has an average width of 4 1/2 feet and an average dip of 75°.

The beds have been displaced 3 to 60 feet by postmineral faults. The faults generally cut the beds at acute angles, and in exploration work they have been mistaken for the footwall of the beds in several instances. Drifts were extended along the faults for considerable distances before the faults were recognized. To minimize the possibility of drifting on these strike faults, one lifter of each drift round is drilled in the footwall waste rock.

In general the beds dip at high angles and are well-adapted to the shrinkage-stopping method of mining. The ore, although hard, breaks well, and the walls of the beds require little support.

Development consists of two inclined shafts (the Stank 730 feet deep and the Humboldt 800 feet deep) with levels at 100-foot intervals. Each shaft is divided into two compartments; the hoisting compartment is 4 1/2 by 4 feet and the manway, 3 1/3 by 4 feet. Shaft sets are 6- by 6-inch timbers, except the hanging-wall plates, which are 6 by 8 inches.

The track for the 1-ton skip in each shaft is 16-pound rails spiked to the footwall plates except at the stations, where they are on 6- by 8-inch stringers placed lengthwise with the dip of the shaft.

Shaft sinking is done on contract, the cost ranging from \$15 per foot for a dry shaft to \$25 per foot for a wet shaft. These prices are for labor only; all supplies are furnished by the company.

¹⁰ Heizer, Ott F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Mines at Mill City, Nev.: Information Circ. 6284, Bureau of Mines, 1930, 13 pp.

The bucket used in sinking has a capacity of 600 pounds. The self-dumping arrangement of the bucket is shown in figure 3. The bucket skids are 4- by 6-inch timbers placed 16 inches apart. The lugs attached to the side of the bucket do not touch the skids except at the dumping place, where the width between the skids is increased to allow the lugs to drop into the notches cut in the skids. The T-shaped arms permit the lugs to slide under the arms when the bucket is raised and over them when the bucket is lowered. The bucket is operated in the manway side of the shaft. To protect the men working in the bottom of the shaft a bulkhead of round timbers is placed below the level. A small, portable, double gear reduction hoist with motor inside the drum is used in sinking.

The various types of rounds used in shaft sinking, raising, and stoping are shown in figure 4. As the ground is unusually hard to drill, Leyner machines are largely employed in shaft sinking, stoping, and drifting. For drilling 1 1/4-inch, hollow, round steel is used with double-taper crossbits. Drill steel containing 2 percent vanadium has been found to give the most satisfactory service. In all blasting 40-percent gelatin dynamite and both no. 6 electric delay detonators and fuse are used.

Drifts and crosscuts are driven 5 1/2 by 7 feet in section and are un-timbered. The development work is so planned that most of the waste rock can be disposed of underground for filling mined-out stopes. Drifts and crosscuts usually are driven on company account; exceptionally wet places may be contracted at a rate of \$5 per foot for breaking and shoveling.

Raises are 5 by 7 feet in section, and virtually all raising is done in ore. Manways and ventilation raises are timbered with stulls placed on 5-foot 4-inch centers. The stulls are lined with 2-inch planks. The timbered raises are of the single-compartment type, and each is equipped with a ladderway and timber slide. Supplies are hoisted to the top of a raise by means of a pulley and hemp rope. The pulley is chained to a length of drill steel placed over the stulls at the top of the raise.

Raises for ore chutes are not timbered. The chute raises are driven to a height of 6 feet above the back of a drift at intervals of 15 or 20 feet and then belled out to connect with each other. The timber chute shown in figure 5 is built in each raise before the bellling-out process is started. In driving chute raises temporary staging is used when necessary.

The ore is mined by the shrinkage-stoping method without filling, except for disposing of waste from development work. The bed is mined in sections, each stope being 75 to 80 feet long. A stope section is shown in figure 6.

To prepare a stope for mining, the bed is first exposed by drifting. A ventilation raise, which also serves as a manway, is driven to the level above as far from the shaft as possible. While the ventilation raise is being driven chute raises are started at intervals of 15 to 20 feet and the chutes installed. A manway is started 80 feet from the ventilation raise if the chute spacing is 20 feet and 75 feet if the chute spacing is 15 feet. The interval is determined by the character of the ore; if the ore is dry and

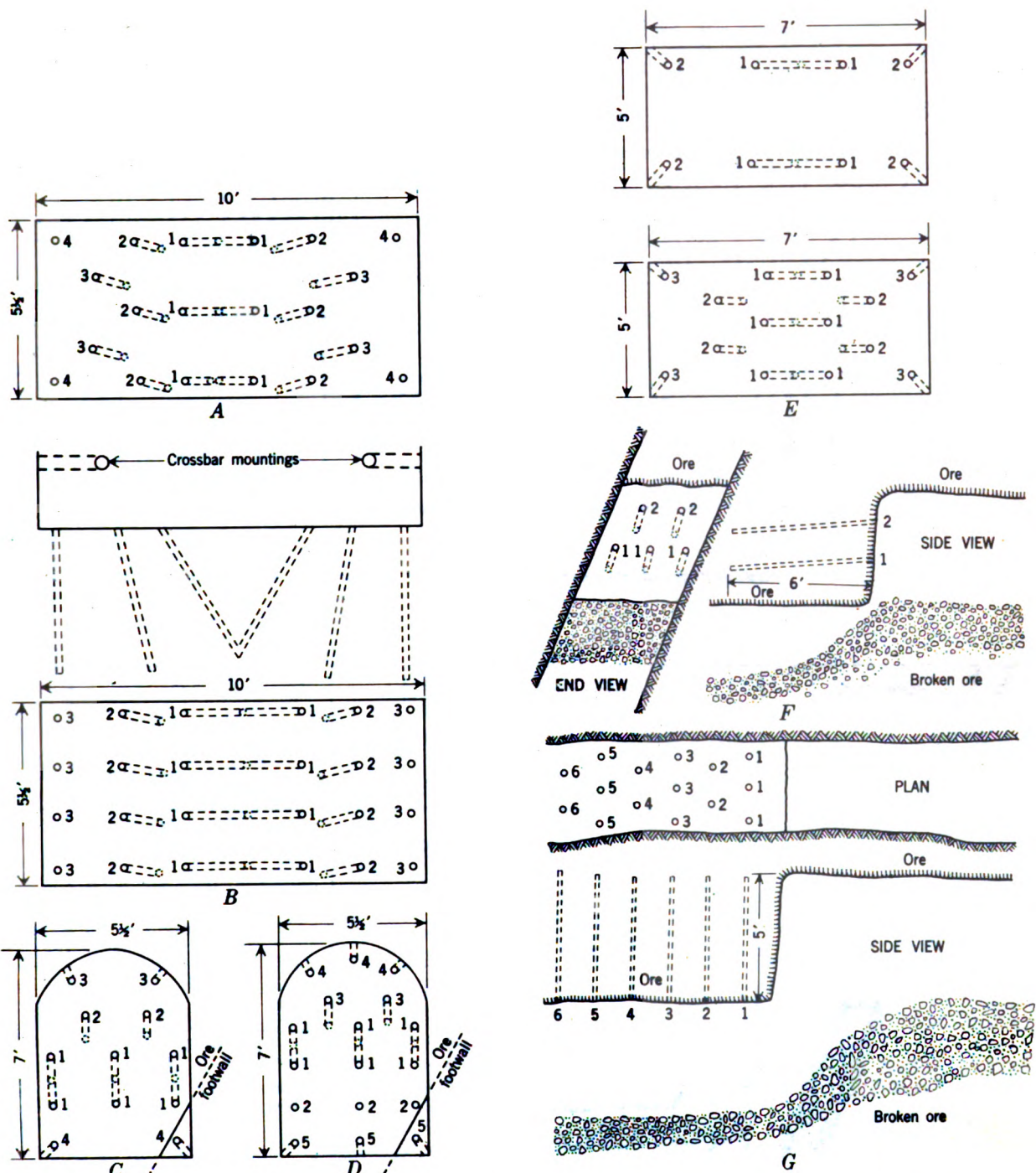


Figure 4.—Drill rounds used at the Nevada-Massachusetts mines. Numbers indicate order of firing: A, 20-hole shaft round; B, 24-hole shaft round; C, 12-hole drift round; D, 17-hole drift round, same round used for crosscuts; E, 14-hole raise round; F, 5-hole stope round in hard ore; G, 15-hole stope round in soft ore.

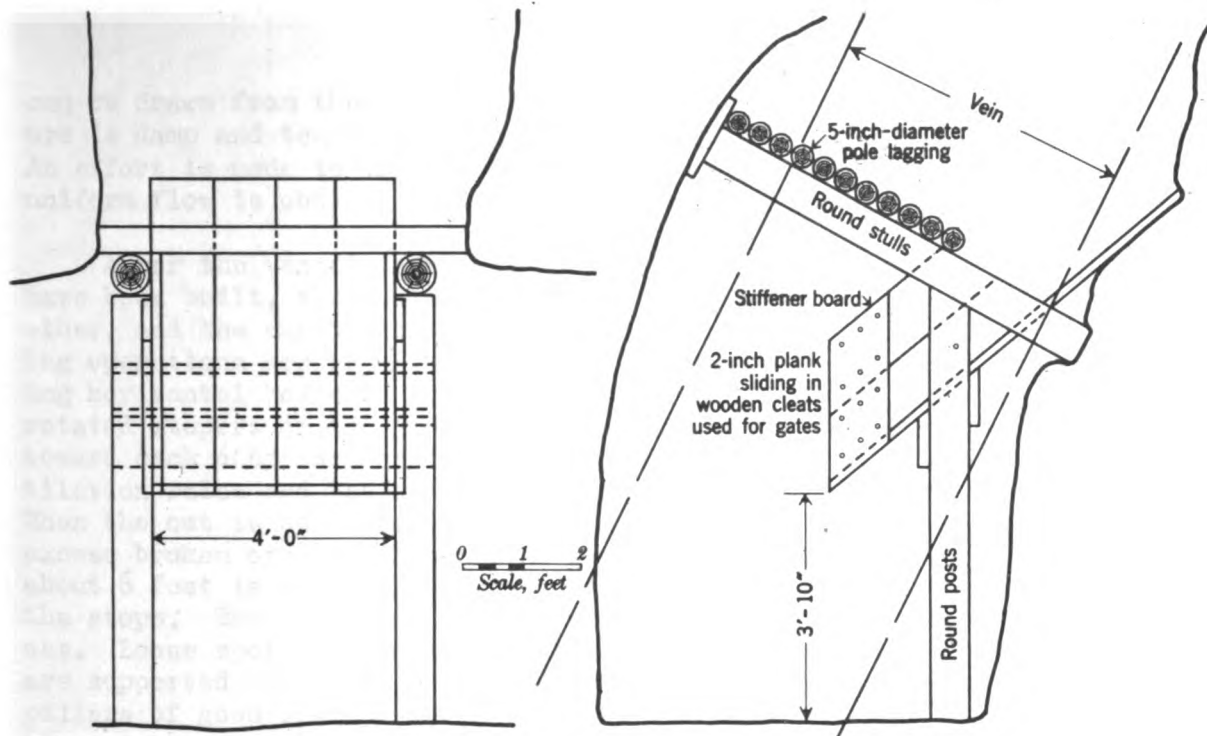


Figure 5.—Sketch of standard chute timbering used at Nevada-Massachusetts mines.

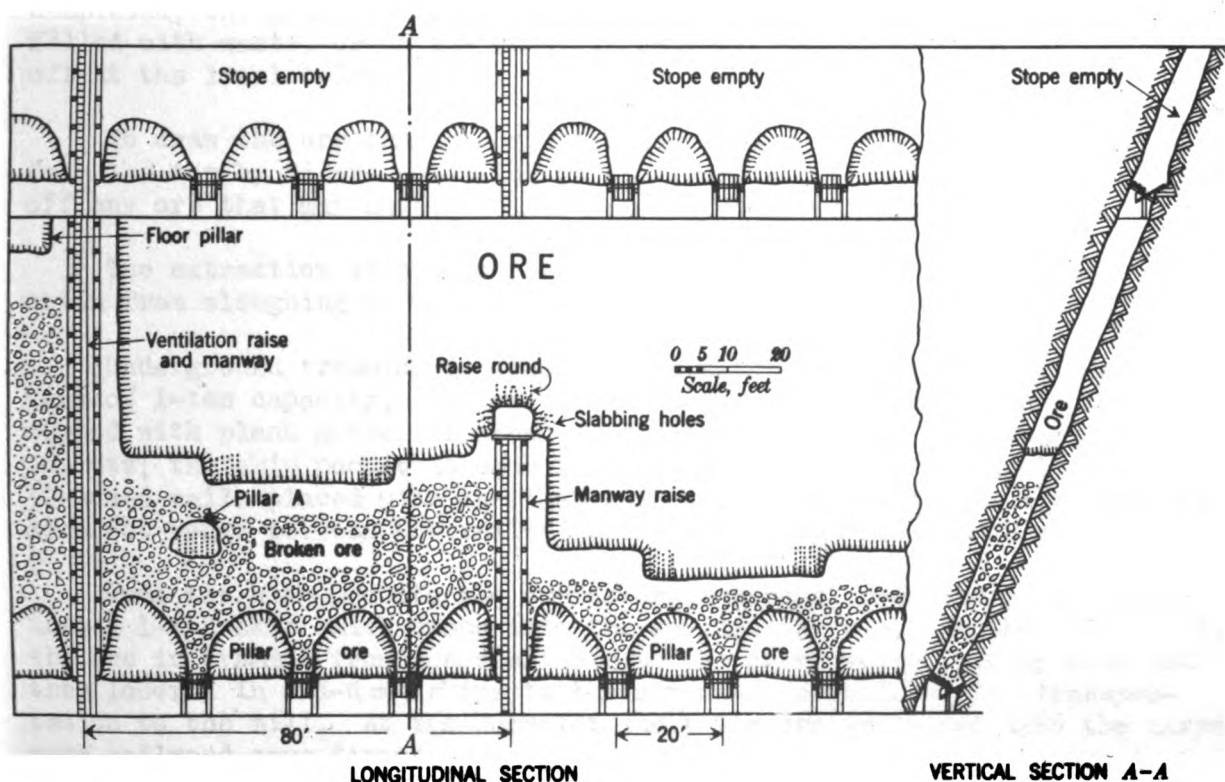


Figure 6.—Longitudinal and vertical sections of shrinkage stope at Nevada-Massachusetts mine.

can be drawn from the chutes readily the 20-foot spacing is used, but if the ore is damp and tends to chimney or hang up the 15-foot interval is chosen. An effort is made to draw the excess broken ore from the stopes uniformly; uniform flow is obtained with damp ore by close spacing of the chutes.

After the ventilation raise has been driven to the level above, the chutes have been built, the chute raises have been belled out to connect with each other, and the manway at the other end of the stope has been started, stoping operations are begun. The stope is advanced as a single face by drilling horizontal holes if the ground is hard or vertical holes with hand-rotated stopers if the ground is comparatively soft. Two miners advancing toward each other are employed in a stope section, one starting at the ventilation raise and the other at the manway at the other end of the stope. When the cut is completed other slices are taken in the same manner. The excess broken ore is drawn off as stoping progresses, so that a distance of about 6 feet is maintained between the top of the broken ore and the back of the stope. The manway is carried upward from time to time as stoping advances. Loose spots encountered in the hanging wall as the stope is advanced are supported either by stulls or by pillars of low-grade ore. Occasionally, pillars of good ore may be left in the stope for support, and these are blasted later when the stope is completed and the ore is being withdrawn.

The stope is advanced to within 6 feet of the floor of the level above, under which a floor pillar is left until the operations on the level are completed; the pillar is then removed by underhand stoping if the stope is filled with waste, or if the stope is empty it is blasted and the ore drawn off at the level below.

To draw the ore from a completed stope, the ore is followed downward from the top by miners who catch up any loose spots in the walls and trim off any ore that may have been left as the stope was carried upward.

The extraction of ore is estimated to be about 95 percent. The dilution from sloughing of wall rock is about 20 percent.

Underground tramming is done by hand with 16-cubic feet, roller-bearing cars of 1-ton capacity. The ore is drawn from the stopes through chutes equipped with plank gates, as shown in figure 5. The ore is dumped into skip pockets; the skip pocket at each level is equipped with a grizzly made of 30-pound rails placed with the flat side up, so that the oversize material does not wedge between the rails. Grizzly openings are 8 inches.

From the skip pockets the ore is drawn through rack-and-pinion gates into a 1-ton skip which dumps into the surface ore bin. At the Stark shaft the ore is trammed from the shaft bin to the gravity-tram bin by hand and then lowered in end-dump skips to the narrow-gage railroad for transportation to the mill. At the Humboldt shaft the ore is loaded into the narrow-gage railroad cars direct.

The track in the mines is made of 16-pound rails laid with an 18-inch gage on a grade of $3/4$ percent.

I. C. 6852

The ore is hauled to the mill by an 8-ton gasoline locomotive. The cars have a capacity of $67 \frac{1}{2}$ cubic feet or about 5 tons each. A train comprises 5 cars. The cost of transportation from the mines to the mill is $3 \frac{1}{2}$ cents per ton-mile excluding depreciation.

Mining costs for 1928 are shown in tables 7 and 8. In 1934 mining costs were about 20 percent less than those given for 1928. Production in 1934 was at a rate of approximately 200 tons per day.

TABLE 7. - Mining costs per ton of ore, Nevada-Massachusetts Co., Inc., Mines
(Tons of ore hoisted: 40,924.1/ Period covered: 1928)

	Labor	Super- vision	Drills & steel	Power	Explo- sives	Timber	Other supplies	Total
Development.....	\$0.557	\$0.012	\$0.025	\$0.092	\$0.100	\$0.036	\$0.085	\$0.907
Mining.....	1.662	.064	.100	.209	.355	.146	.207	2.743
Pumping.....	.050	---	---	---	---	---	.055	.105
Transportation ^{2/}	.053	---	---	---	---	---	.137	.190
Total.....	2.322	.076	.125	.301	.455	.182	.484	3.945

1/ About 15,000 tons of ore were broken but not hoisted during 1928.

2/ Surface tram to mill; underground tramping included in mining and development.

TABLE 8. - Mining costs in units of labor, power, and supplies,
Nevada-Massachusetts Co., Inc., mines, 1928

A. Labor (man-hours per ton):	Development	Mining	Total
Breaking.....	0.183	0.700	0.883
Timbering.....	.053	.376	.429
Shoveling and tramping.....	.167	.383	.550
Shaft sinking.....	.098	---	.098
Hoisting.....	.136	.552	.688
Supervision.....	.040	.168	.208
Pumping.....	---	---	.195
Total	.677	2.179	3.051
Average ore mined per man-shift (tons).....			2.62

3. Power and supplies:	Development	Mining	Total
Explosives, pounds per ton.....	0.316	4.648	4.964
Timber, board-feet per ton.....	.73	2.90	3.63
Poles, lin. ft. per ton....	.12	.49	.61
Power, kw.-hr. per ton.....	3.25	7.23	10.48

	percent of total direct cost
C. Labor.....	56.02
Power and supplies.....	43.98

CONCENTRATION OF TUNGSTEN ORES

Introduction

During the past 20 years at least 60 concentrators, ranging in capacity from several tons to 300 tons per day, have been erected in the United States for treating tungsten ores. Many of these mills were erected hurriedly during the World War and operated for only a short time when the demand for tungsten as a pivotal war mineral was imperative. Frequently these hastily designed and built mills were able to operate only because of the extremely high prices paid for tungsten concentrates at that time.

Concentration of tungsten ores in the United States depends basically on gravity methods. Under normal conditions tungsten concentrates must contain at least 60 percent WO_3 and be virtually free from such impurities as sulphur, phosphorus, arsenic, antimony, bismuth, copper, tin, and molybdenum ordinarily associated with tungsten ores; consequently gravity concentration and auxiliary methods must be designed to produce a very clean concentrate.

Crushing and Grinding

Since grinding is the chief item of cost in concentration and since the method of grinding has an important bearing on recovery, this phase of concentration should receive careful consideration. In the early milling of tungsten ores the tendency was to employ grinding equipment originally designed for ores of other metals, namely, gravity stamps and to a smaller extent ball mills. In the light of present knowledge it is not surprising that the efficiency of these early tungsten mills was low. Recoveries of 60 percent were considered good.

Tungsten minerals slime readily in a stamp battery because they are brittle and their high specific gravity prevents their discharge from the battery unless they are finely divided. They also slime excessively in a ball mill as usually operated, thus rendering their concentration difficult. Grate-type ball mills, however, have been employed successfully at the Round Valley and Pine Creek tungsten deposits near Bishop, Calif. These mines avoid excessive slining by using large quantities of water in the ball mills and carrying the ore through them quickly.

Stamps for crushing tungsten ores have become virtually obsolete; present practice favors the use of primary crushing in Blake-type crushers, followed by stage crushing with two or more sets of rolls. Where the tungsten mineral is associated intimately in a cryptocrystalline aggregate of quartz and ferberite, as in some of the Boulder (Colo.) ores the ore is ground in grate-type ball mills after the coarser products have been recovered.

Classification

To obtain a good recovery from tungsten ores they must be classified thoroughly before being concentrated. The fundamental principle in modern milling practice involves grading the product according to size as it passes through the mill and eliminating as much of the tungsten minerals as possible from the various sizes by machines best adapted for their recovery. This procedure is important in that (1) it gives a higher recovery by preventing slime loss in subsequent regrinding operations and (2) it reduces the amount of equipment necessary for treatment of slime if such equipment is employed. Operators realize the importance of making a clean sand-slime separation so that these products can be treated on separate machines. As in all gravity concentration, slime treatment presents the most difficult problem. The best practice is to reduce the amount of slime to a minimum, separate it from the sand, and treat it in as long a series of machines as is consistent with economy.

Jigging

The jig is very useful in concentrating tungsten ores if the tungsten mineral occurs in coarse form in that it permits the recovery of a coarse, high-grade concentrate, thereby decreasing the loss of slime. Opinion is divided as to the relative merits of the several types of jigs. The fixed-screen, Harz-type jig, however, has been adopted more generally. With this type of jig a large amount of ore can be retained on the screen over a long period, and accumulations on the screen can be adjusted. The use of jigs in tungsten concentration tends toward compactness, since the space required for subsequent table treatment is reduced considerably.

Tabling

Various makes of tables of the vibratory type, such as the Card, James, Isbell, Deister, Wilfley, and Overstrom, have given good service in the concentration of tungsten ores. Although some mills employ two or more makes of tables it may be advantageous to use only one, or at most two, in order to gain the benefit of standardization of equipment.

Ample table capacity should be provided because generally the extraction is reduced decidedly if the tables are overloaded.

Slime Treatment

Various types of equipment - Monell tables, Frue vanners, and stationary canvas tables or "rag" plants - have been used to treat slimes produced in tungsten milling. The most favored slime-concentrating devices were the canvas tables, which were used most in the Boulder district during the World War. Before the canvas tables were installed in this district the tailings were sent to waste, although they carried values which could be recovered partly by the use of the canvas tables. In present practice canvas tables have been replaced largely by tables of the vibratory type. The objection to the use of a canvas plant is the difficulty of marketing the resulting low-grade concentrates, which range from 15 to 35 percent WO_3 . Production

of a high-grade concentrate was not so essential during the World War, when concentrates of almost any grade could be sold readily at a good price.

Dewatering

Mechanical thickeners generally are used in tungsten milling for dewatering the pulp. Dewatering not only enables the pulp to be handled more efficiently but is also an economical practice, particularly in districts where water is difficult to obtain. By the use of dewatering equipment part of the mill water can be reclaimed for further use.

Roasting

When considerable pyrite is present in tungsten ores clean separation of the pyrite and tungsten minerals by gravity methods is not economically feasible. The scheme of treatment usually followed is to allow much of the pyrite to concentrate with the tungsten minerals, then to roast the concentrate and remove the iron by means of a magnet. The bulk concentrates are dried before roasting. The driers employed are either of stationary or rotary types. The drier used in the Boulder district is an ordinary stove with a grate for fuel; it is surrounded by a sheet-iron jacket perforated at the bottom. (See fig. 7.) The wet concentrates are charged into the jacket and come in contact with the hot stove; when dried they are discharged through the perforations at the bottom.

Pyrite is separated from tungsten concentrates by a magnetizing roast followed by magnetic separation. The pyrite must be present in individual grains for complete separation; the more grains the more effective the separation by magnetic means. The thoroughness of the magnetizing roast is another factor. The object of roasting is to form a magnetic film that will adhere to the particle.

The important factors in roasting are the size of the particles, the temperature, and the time.¹¹ Coarse particles require appreciably longer to heat to ignition temperature than fine ones. A temperature at least equal to the ignition temperature of pyrite must be maintained long enough for a film to form. The time of roasting need not be prolonged beyond this point. If the temperature is high enough the film will form in a few moments.

The physical appearance of the roasted product is a guide to roaster operation. A good magnetizing roast is indicated by the absence of characteristic pyrite particles and by a flat, dead color ranging from bronze to black. Black is characteristic of a deficiency of air during roasting or of a reducing atmosphere. The color of the roasted product is affected by the amount and color of other minerals present, which may or may not be altered in roasting.

¹¹ Young, George J., Experiments in Magnetizing Roasting: Eng. and Min. Jour., vol. 108, 1919, pp. 176-177.

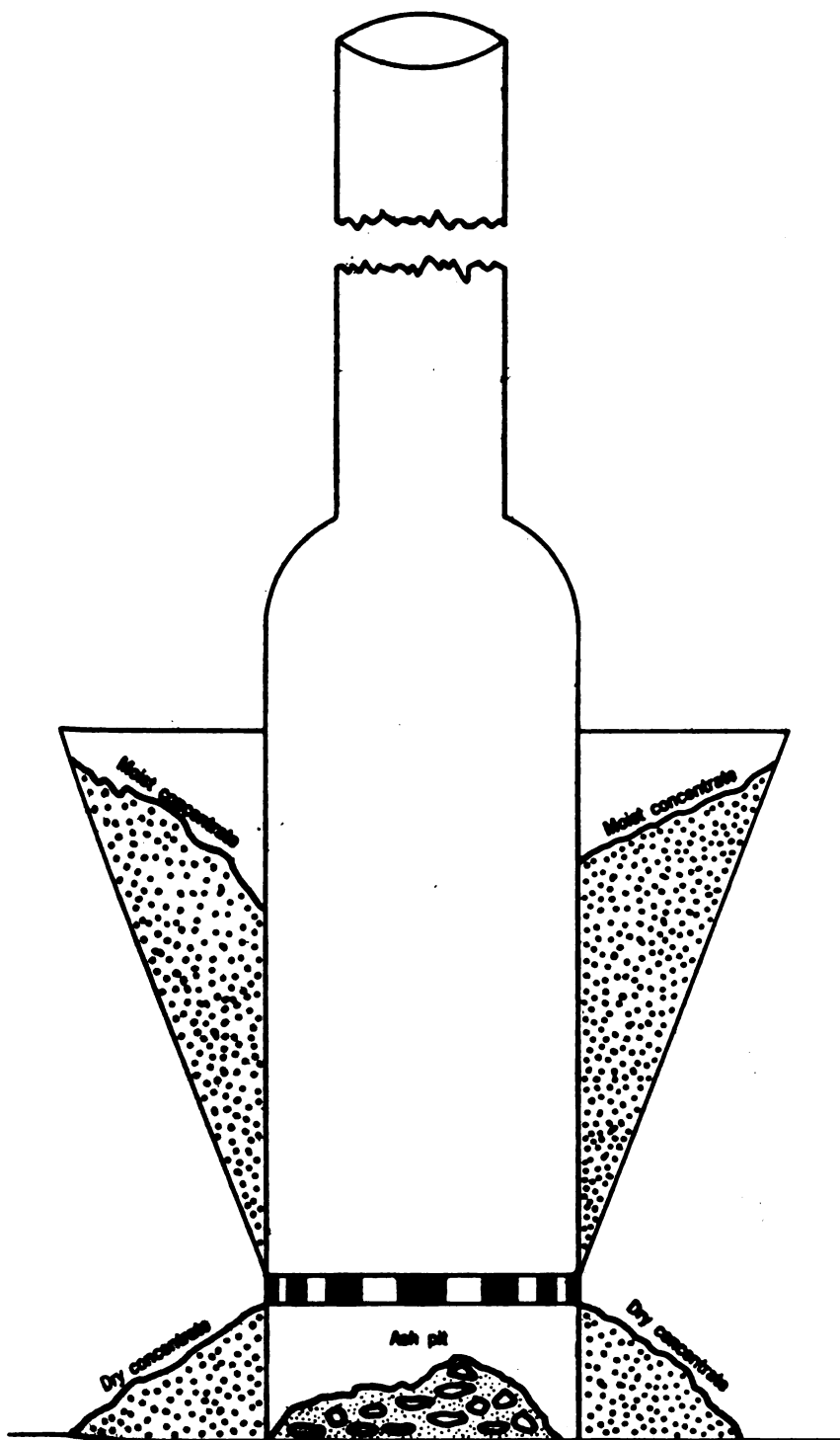


Figure 7.—Sketch of homemade concentrate drier used in the Boulder district.

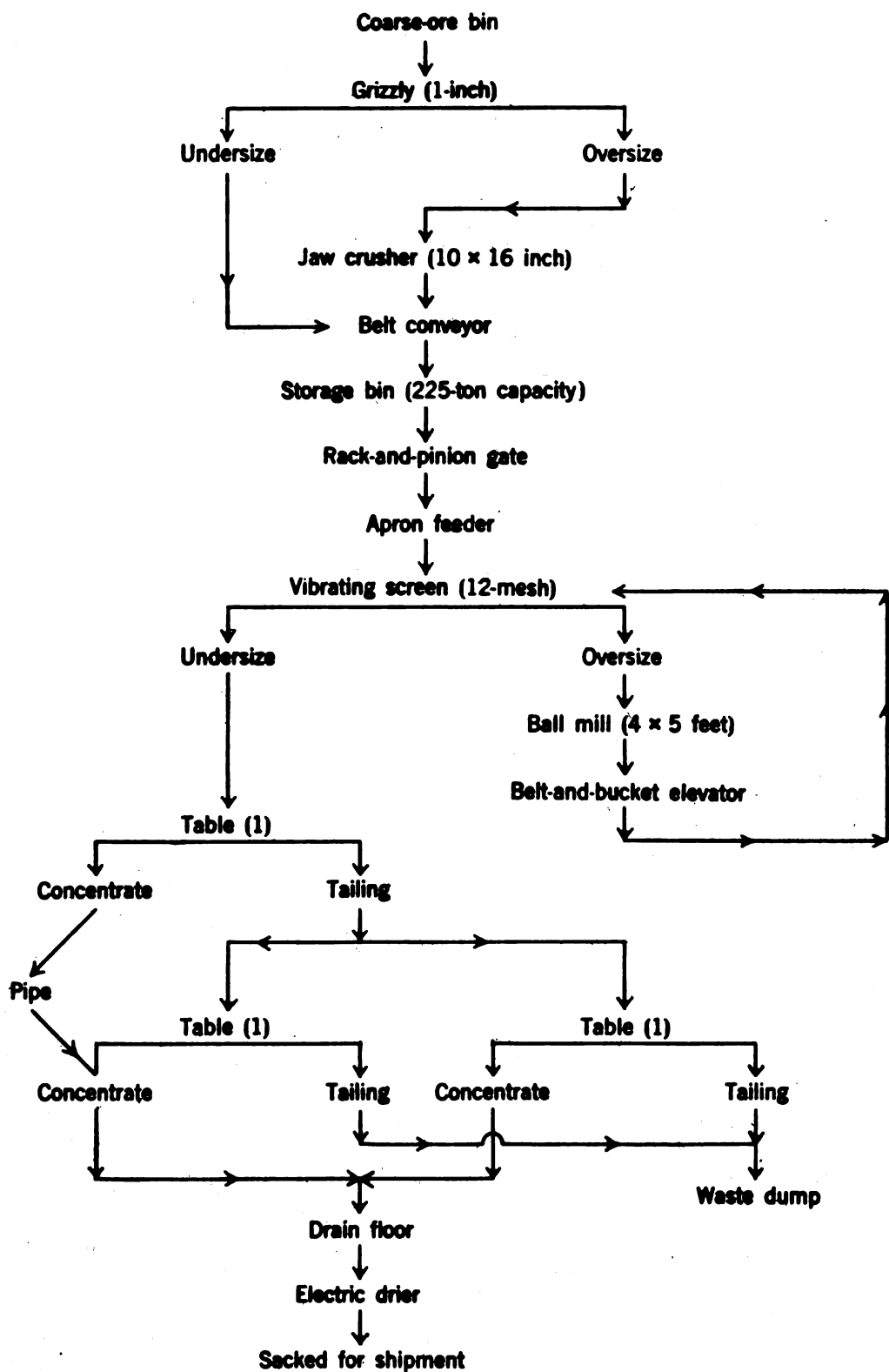


Figure 8.—Flow sheet of Round Valley tungsten concentrator.

A roasting furnace of any type can be used providing the concentrates are passed through quickly enough. Furnaces of the rotary type made of iron pipe and heated with oil give satisfactory service.

Magnetic Separation

Wetherill or Ding's magnetic separators have proved useful adjuncts to table concentration when the ores contain pyrite, garnet, and epidote. With tungsten ores of the contact-metamorphic type, which usually contain all three gangue minerals, the plan generally favored is to produce a bulk concentrate by tabling. After the concentrates are dried the garnet and epidote are removed by magnetic separation. The concentrate containing the pyrite is then roasted to convert the pyrite into iron oxide, as previously described, when it can then be removed by magnetic separation.

Some tungsten ores, such as those of Cornwall, England, contain both wolframite and tin; inasmuch as the specific gravities of these two minerals are nearly the same considerable difficulty is experienced in making a separation either by wet or dry concentration. ¹²The advent of the magnetic separator did much to mitigate this difficulty. After the mixed concentrates from tables or buddles have been roasted to expel sulphur and arsenic they are sent to a magnetic separator, where the iron is removed first by a weak magnet and then the wolframite by a strong magnet. One difficulty was the fact that the wolframite concentrate contained considerable tin cemented by iron oxide. At one mine the oxide of iron was dissolved by hot sulphuric acid, and the freed wolframite particles were separated from the tin by being passed through magnetic fields of varying intensity.

Round Valley Tungsten Co. Concentrator

The concentrator of the Round Valley Tungsten Co. near Bishop, Calif., has a maximum capacity of 120 tons of scheelite ore per day. A plentiful supply of water for milling is available from Horton Creek in the vicinity. Power is purchased from the Southern Sierras Power Co. under a variable schedule, the average cost being about 1 1/2 cents per kilowatt-hour.

The ore contains an average of 0.5 percent WO_3 , with garnet, epidote, and quartz as the principal gangue minerals. The scheelite crystals range from 1/32 to 1 inch in diameter.

As the grade of the ore is low the scheme of treatment must necessarily be simple. The flow sheet of the concentrator is shown in figure 3.

The broken ore from the glory hole is hoisted and dumped direct into the crusher bin, which is equipped with a plank gate to control the flow of ore into the 10- by 16-inch Blake-type crusher. A 40° inclined bar grizzly with 1-inch openings is ahead of the crusher. The minus 1-inch product from the crusher and the undersize from the grizzly drop onto a conveyor belt,

¹² Taylor, M.T., Separation of Wolfram from Tin: Min. Mag., vol. 12, 1915, pp. 351-352.

which discharges into the 225-ton storage bin.

From the bin the ore is fed through a rack-and-pinion gate onto an Allis-Chalmers apron feeder 24 inches wide. It discharges the ore onto a 12-mesh vibrating screen, which removes about 12 percent of the total ore in fines before it reaches the ball mill. The screen oversize passes into a 4- by 5-foot Marcy ball mill equipped with 3/16-inch grate openings. Excessive sliming in the ball mill is prevented by adding about 6 tons of water to 1 ton of ore. The ball-mill discharge goes to a belt-and-bucket elevator, whence it is returned to the 12-mesh screen. The screen oversize is returned to the ball mill. The large circulating load, mixed with a large volume of water, prevents sliming; screen tests show that only about 14 percent will pass a 150-mesh screen.

Forged chrome-steel balls 5 inches in diameter are used in the ball mill. The load of balls in the mill is about 3 1/2 tons; ball consumption is 0.9 pound per ton of ore. Ball-mill liners last about 9 months when the mill is operated continuously.

The screen size for table concentration is minus 12-mesh. Tests have shown that this size liberates most of the scheelite and permits the mill to handle a large tonnage. The undersize from the 12-mesh screen is run to an 18- by 5-foot Overstrom table, which makes a finished concentrate and a tailing. The tailing from this table is split into two parts, which are treated on two additional Overstrom tables, thus making concentrates and finished tailings. About 90 percent of the recovered scheelite is obtained from the first table.

The table concentrates are collected by hand and placed in a pile on a cement floor to drain for 12 hours. After draining they are shoveled onto an electric drier. (See fig. 9.) The drier is home-made and consists of nine 2-inch pipes 6 feet long arranged in parallel; each pipe contains three standard 220-volt space heaters placed in series. These space heaters are connected to the 440-volt circuit. Since three 220-volt heaters are used for each tube the rated capacity of the heaters is reduced considerably, thus giving the heaters longer life and furnishing the right amount of heat to dry the concentrates. The cost of electricity is about \$2.50 per ton of concentrates. The pipes are placed directly on a 10-mesh screen and the damp concentrates shoveled on top. The concentrates pass through the screen when thoroughly dried and fall into the bin below. The drier requires little attention, and there is slight danger of fire. The capacity of the drier is about 2 tons per 24 hours.

The dried concentrates are shoveled into 100-pound canvas sacks covered with jute sacks.

The following tables give costs of milling and distribution of labor and power for a 1-month period at the Round Valley concentrator.

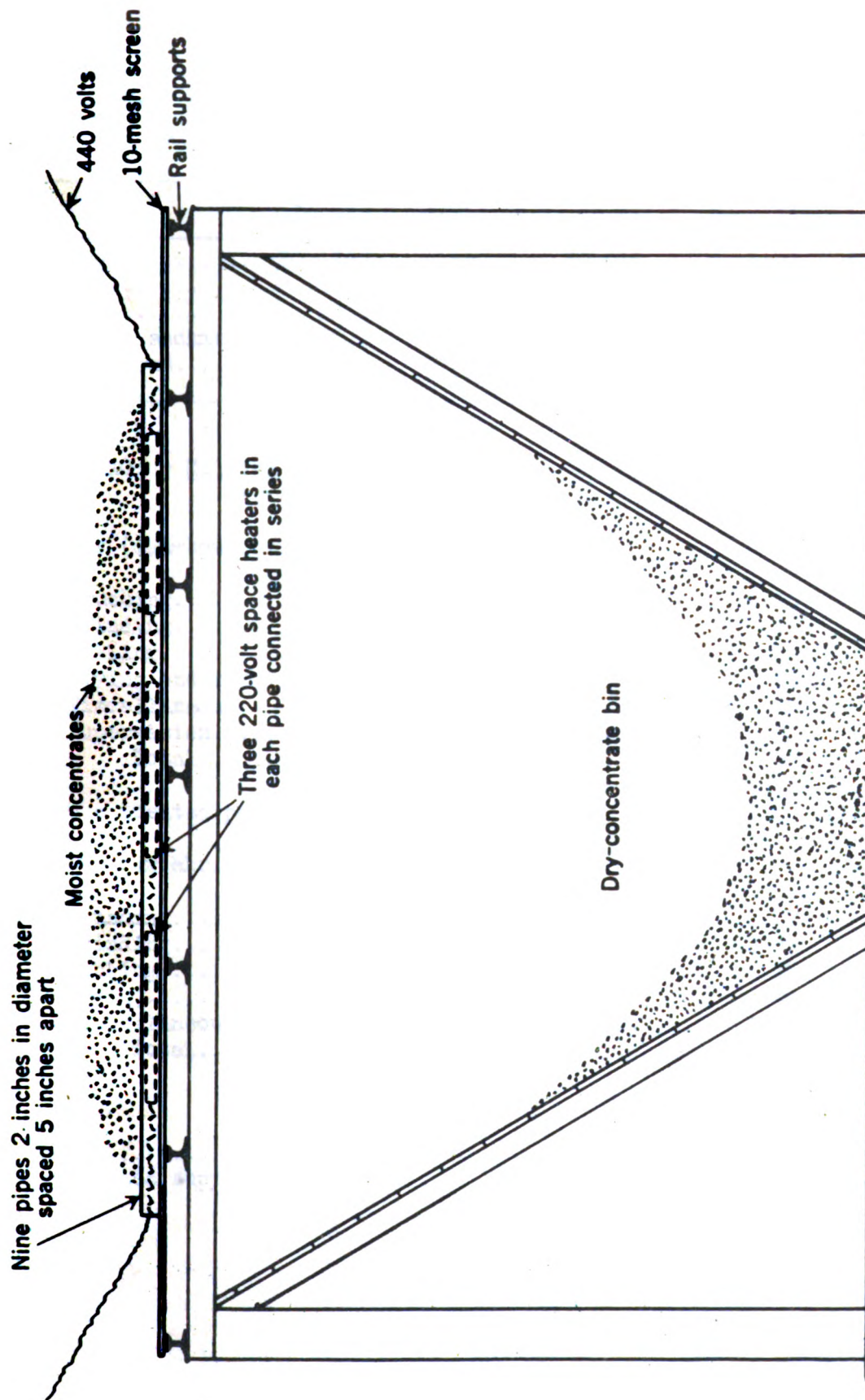


Figure 9.— Sketch of electric concentrate drier used at Round Valley mill.

TABLE 9. - Milling costs per ton of ore, Round Valley concentrator

(Tons ore treated: 2,250. Period: 1 month, 1931)

	Labor	Supervision	Power	Supplies	Total
Crushing.....	\$0.066	---	\$0.016	---	\$0.082
Grinding.....	.100	---	.097	\$0.048	.245
Tabling.....	.100	---	.007	---	.107
Drying and sacking..	.050	---	.006	.147	.203
Miscellaneous.....	.050	\$0.133	.017	.020	.220
Total.....	.366	.133	.143	.215	.857

TABLE 10. - Milling costs per ton of ore in units of labor, power, and supplies, Round Valley concentrator

A. Labor (man-hour per ton):

Crushing.....	0.107
Grinding.....	.160
Tabling.....	.160
Drying and sacking.....	.053
Miscellaneous.....	.053
Supervision.....	.053
Total.....	.586

Ore treated per man-shift (tons).....13.65

B. Power (kw. hr. per ton of ore treated)..... 9.54

Crushing.....	1.07
Grinding.....	6.47
Tabling.....	.47
Drying.....	.40
Miscellaneous.....	1.13
Total.....	9.54

Percent of
total cost

C. Labor.....	58.2
Power and supplies.....	41.8

Atolia Placer-Ore Concentrator

The Atolia concentrator for concentrating scheelite obtained from placer material is at Atolia, Calif. The capacity of the concentrator is 500 tons per 16-hour day.¹³ Water for mill use is pumped from a well 7 miles east of Atolia. This water is highly charged with chlorides and sulphates but is suitable for gravity concentration. Before this source of mill water was developed water was shipped to Atolia in railroad tank cars from wells west of Kramer, 40 miles distant. Electric power is purchased from the Southern Sierras Power Co. at a rate of about 2 cents per kilowatt-hour.

The placer scheelite occurs in alluvium consisting of sand and gravel with few boulders. The alluvium is cemented loosely so that it must be disintegrated before it can be concentrated. A little more than $1\frac{1}{2}$ pounds of scheelite concentrates are obtained per ton of ore.

The unusual feature of the concentrator is the large amount of material handled at low cost. The direct cost of mining and milling for the first 10 months of 1930 was 37 cents per ton. The flow sheet of the concentrator is shown in figure 10.

The placer material is loaded into trucks by power shovel and hauled several hundred yards to the concentrator. The trucks are unloaded onto a grizzly with 6-inch openings placed over the top of a bin. The grizzly oversize is broken by hand sledges.

From the grizzly bin the ore passes to an inclined conveyor belt which discharges into the disintegrator. The disintegrator consists of a trommel screen with 2-inch perforations; a steel shaft to which twelve 16-pound hammers are attached by chains passes through the trommel. (See fig. 11.) The loosely cemented gravel is broken by the revolving hammers, and the screen undersize drops into a wooden bin. From this bin the ore is transported by conveyor belt and bucket elevator to the mill storage bins.

From the mill bins the ore is fed to four 2-compartment Harz-type jigs by hopper feeders and feed belts. The jigs are equipped with 25- by 34-inch screens with 0.03-inch openings. The height of the bed in the jigs is about 10 inches. The plungers are operated with a 5/8-inch stroke at 150 strokes per minute.

The jigs produce concentrates, hutch, and tailings. The concentrates are skimmed from the jigs at the end of every two shifts. The hutch products, which comprise about 25 percent of the mill feed, are collected in a common sump, whence they are pumped to two Overstrom tables. These tables produce concentrates, middlings, and tailings. The concentrates, which contain large quantities of magnetite, are sent to a magnetic separator for removal of the magnetite. The table middlings are returned to the tables, and

¹³ Vanderburg, W.O., Methods & Costs of Concentrating Tungsten Ores at Atolia, San Bernardino County, Calif.: Information Circ. 6532, Bureau of Mines, 1931, 12 pp.

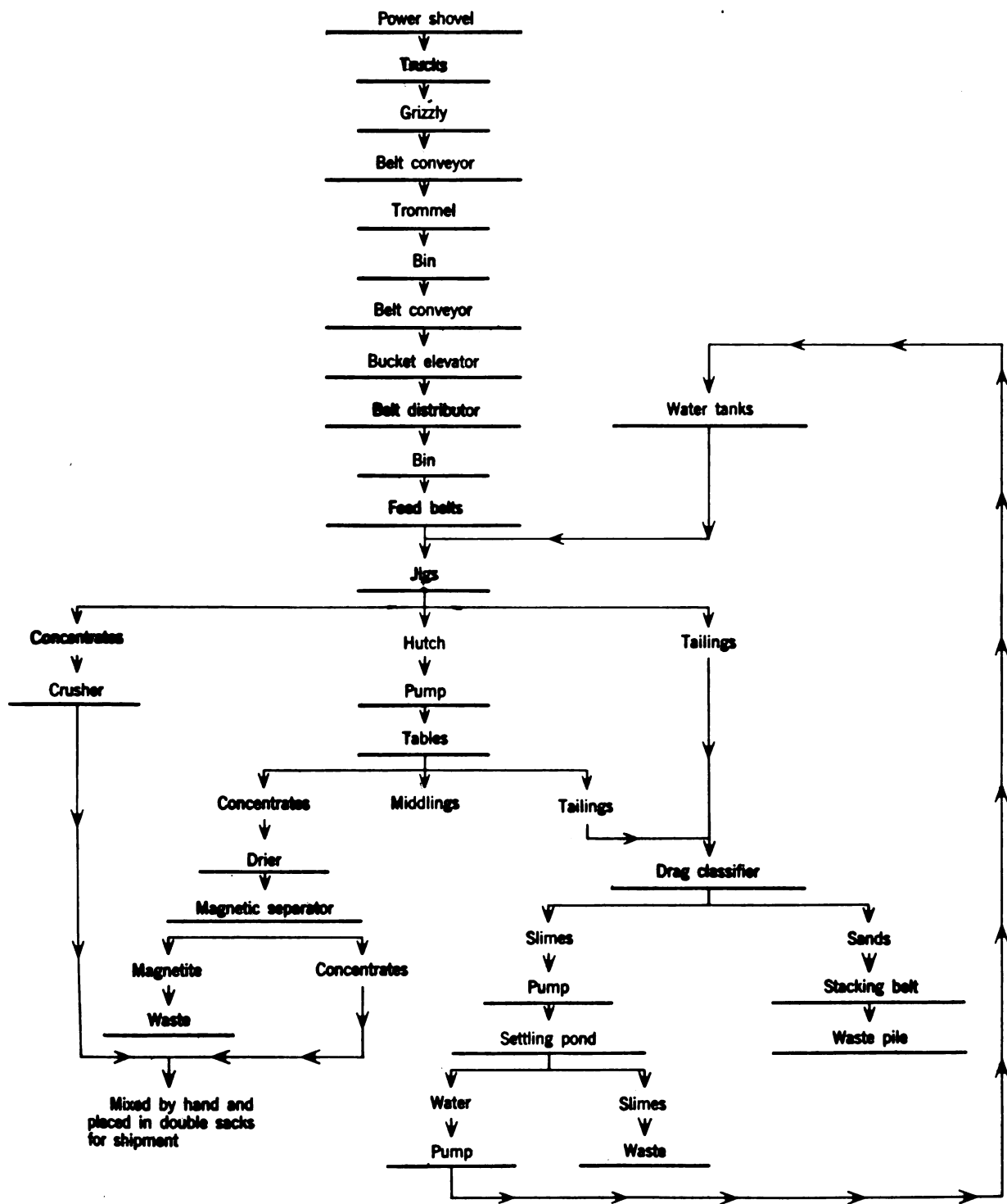


Figure 10.—Flow sheet of Atolia placer-ore concentrator.

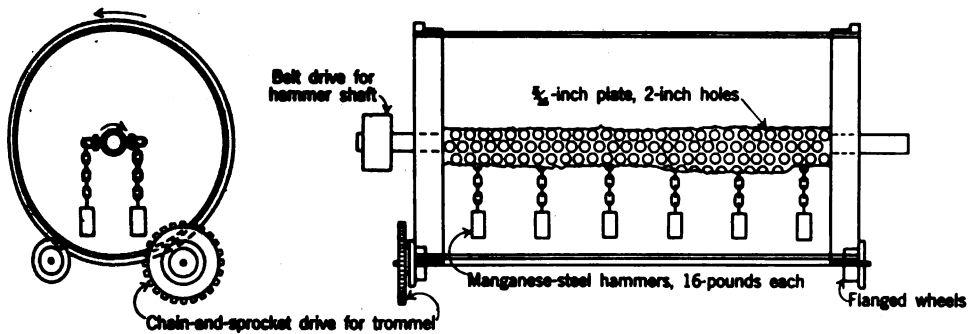


Figure 11.—Sketch of disintegrator for breaking cemented placer material.

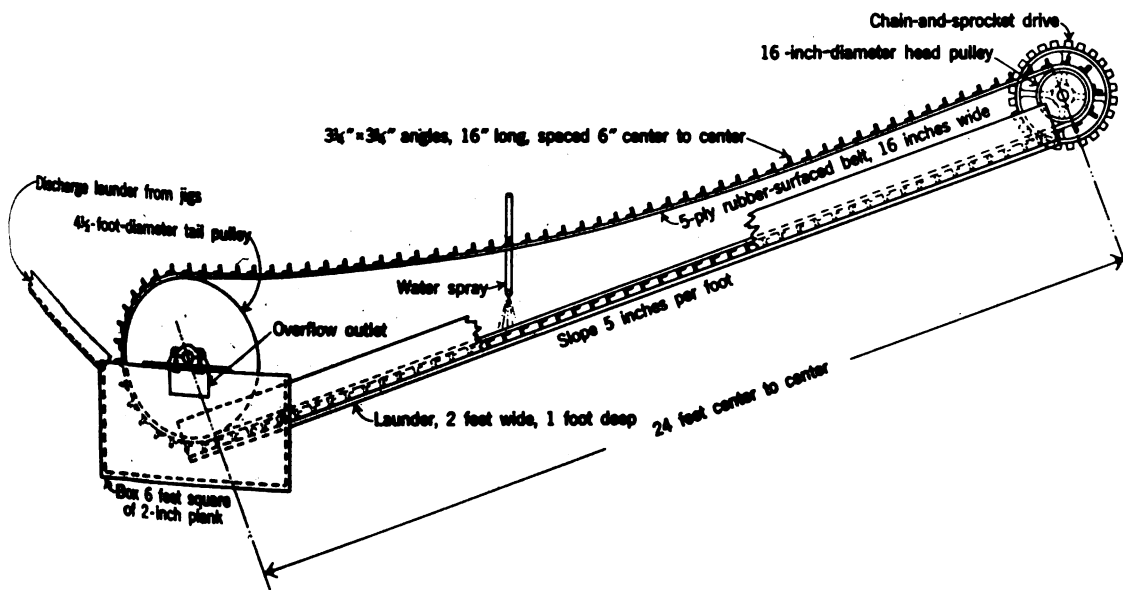


Figure 12.—Sketch of home-made classifier used at Atolia.

the table tailings join the jig tailings for dewatering.

The table concentrates are dried on a wood-fired oven and shoveled into a hopper of the magnetic separator. The magnetic material is removed, and the scheelite is discharged into a wheelbarrow. The jig concentrates are crushed to 3/4 inch in a laboratory Blake-type crusher, mixed with the cleaned table concentrates, and sacked for shipment. The placer concentrates average 62 percent WO_3 , 0.71 percent phosphorus, and a negligible amount of sulphur.

The tailings are dewatered in home-made drag classifiers; these classifiers are equipped with 3 1/4-inch angle irons 16 inches long bolted to 5-ply, rubber-surfaced belts 16 inches wide. The angles are spaced 6 inches apart. (See fig. 12.) The launder in which the drags operate is 12 by 24 inches in section, is made of 2-inch planks, and has a slope of 5 inches per foot. The tailings are discharged from the classifiers with about 12 percent moisture.

The dewatered tailings discharge onto a conveyor belt, which in turn discharges onto a stacking belt the outer end of which is pivoted so that it swings through an arc of 300°.

The overflow from the classifiers flows to sumps whence it is pumped to a settling pond. Clear water is reclaimed from the pond, and it is returned to the supply tanks by pump. The mined-out placer channels are used for settling ponds. The sides of the settling ponds are built up by power shovel.

The recovery of placer scheelite is estimated to be about 70 percent. The losses in concentration are due to the fact that (1) scheelite is mechanically locked with quartz; (2) it is intermixed with placer gravel that has not been disintegrated; and (3) some of it occurs as slime which cannot be recovered entirely by the tables.

The following tables give costs of mining and milling and distribution of labor and power at the Atolia placer operations for a 10-month period.

I.C. 6852

TABLE 11. - Mining and milling costs per ton of ore, Atolia placer-ore concentrator

(Tons ore treated: 106,122. Period covered: 10 months, 1930)

	Labor	Power	Supplies	Total
Mining and trucking.....	\$0.040	—	\$0.060	\$0.100
Milling.....	.127	\$0.040	.030	.197
Water supply.....	.030	.020	.025	.075
Total	.197	.060	.115	.372

TABLE 12. - Mining and milling costs per ton of ore in units of labor, power, and supplies, Atolia placer-ore concentrator

A. Labor (man hour per ton):

Superintendence.....	0.020
Trucking.....	.039
Shoveling.....	.020
Grizzlymen.....	.015
Screening.....	.020
Conveying.....	.006
Jigging.....	.062
Tabling.....	.018
Dewatering.....	.012
Maintenance.....	.020
Pumping.....	.042
Magnetic separation and sacking.....	.008
Total.....	.282

Ore treated per man-shift (tons)..... 23.4

B. Power (kw.-hr. per ton of ore):

Conveying.....	0.995
Screening.....	.315
Jigging.....	.103
Tabling.....	.103
Dewatering.....	.370
Pumping.....	.103
Water supply.....	1.047
Magnetic separation.....	.021
Total.....	3.057

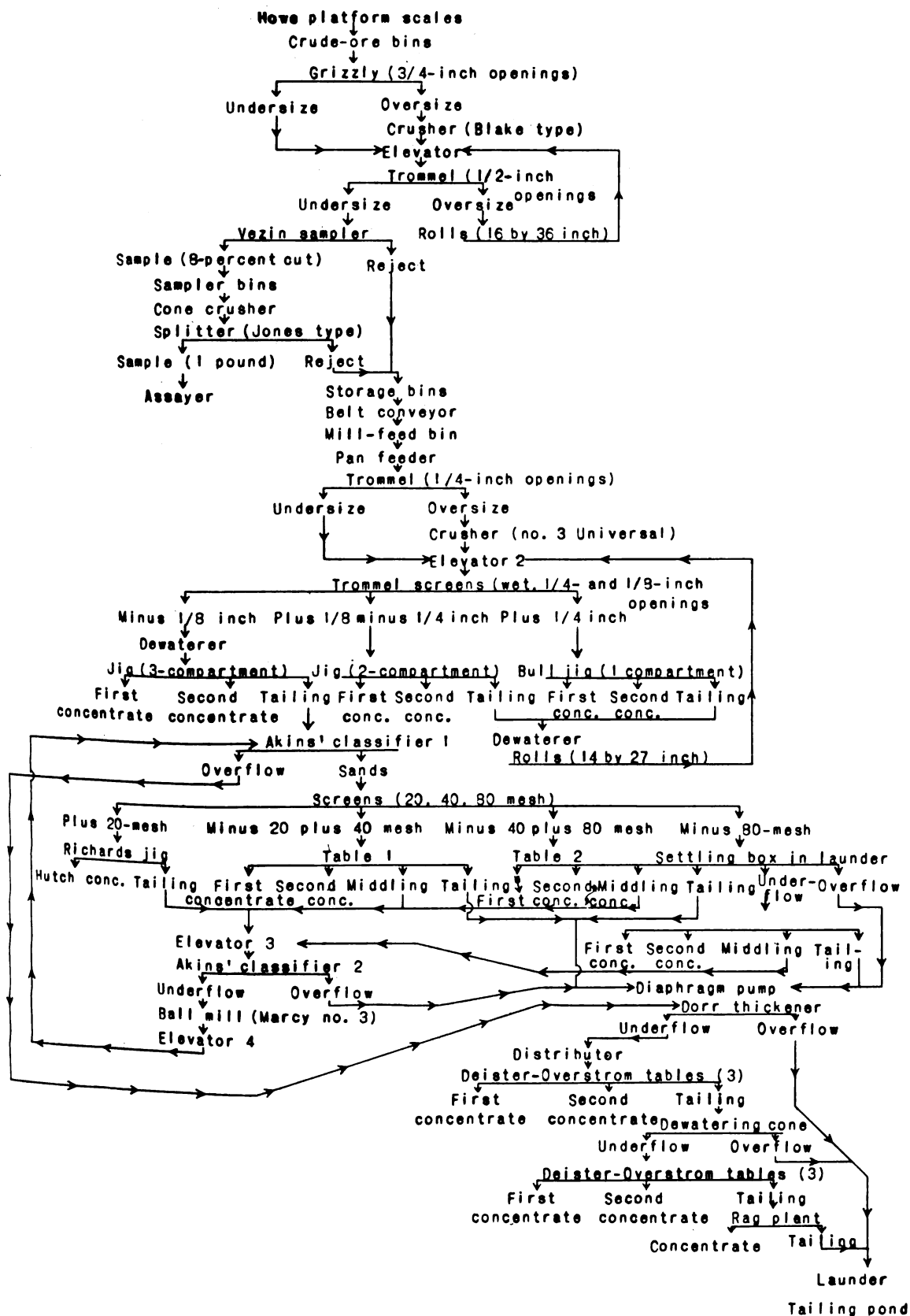


Figure 13.- Wolf Tongue concentrator flow sheet.

Wolf Tongue Concentrator

The Wolf Tongue concentrator is on Middle Boulder Creek at Nederland, Colo. ¹⁴ The concentrator has a maximum capacity of 1 ton of ore per hour when treating ferberite ores containing about 4 percent WO_3 . With higher-grade ores the capacity is reduced proportionately and may be as low as 1/2 ton per hour with ores containing more than 10 percent WO_3 . The total cost of the mill fully equipped was \$35,000, of which \$60,000 was the contract price for the building and new machinery and the balance was for reconditioning and setting up the old machinery damaged by fire in 1927.

The company owns a water right on Middle Boulder Creek whence power for milling is generated by a Leffel wheel direct-connected to a 100-kva generator. Since plenty of water is available for milling there is no necessity for reclaiming the used mill water.

The only mineral in the ore of economic importance is ferberite. The ore treated at the Wolf Tongue mill in 1931 averaged 5.72 percent WO_3 . The principal gangue mineral is quartz.

The ores treated may be classified into two general types: (1) Ores in which the ferberite is liberated from the gangue by coarse crushing and (2) ores of the hornstone or chalcedonic type in which the ferberite and quartz comprise a cryptocrystalline aggregate requiring fine grinding to liberate the ferberite. Because of the extreme friability of ferberite a certain amount of sliming is unavoidable, particularly with type (2) ores. Although the specific gravity of ferberite is 7.5 and that of the gangue about 2.7, no gravity-concentration equipment has been wholly effective in recovering slimed ferberite as a high-grade concentrate. In milling a compromise must be made between the grade of the concentrate and the recovery. Adjustment of the equipment to produce a high-grade product results in a low recovery, and high recovery results in a low-grade concentrate which is difficult to market. Recoveries from different ores, ranging from 3 1/2 to 30 percent WO_3 , have averaged about 90 percent, although some of the concentrates produced were low grade. Owing to the large loss in slimes it has been found unprofitable to treat ores containing less than 3 percent WO_3 by gravity methods under present conditions.

Milling practice at the Wolf Tongue mill is distinguished by the large amount of concentrating equipment required to treat a comparatively small tonnage of ore and by the meticulous attention given the concentrating equipment by the mill operators. The mill flow sheet is shown in figure 13.

The ore is first weighed on a Howe platform scale and then dumped from trucks into bins, whence it passes over an inclined-bar grizzly with 3/4-inch openings. The grizzly oversize is crushed to 3/4 inch by a Blake 7- by

¹⁴ Vanderburg, W.O., Methods & Costs of Milling Ferberite Ore at the Wolf Tongue Concentrator, Nederland, Boulder County, Colo.: Information Circ. 6635, Bureau of Mines, 1933, 15 pp.

10-inch crusher. The crusher discharge and the grizzly undersize pass by gravity to a belt-and-bucket elevator which discharges into a screen trommel with 1/2-inch openings.

The trommel oversize is crushed to 1/2 inch by a set of 16- by 36-inch McFarlane rolls. The roll discharge is returned to the 1/2-inch trommel by elevator 1.

The 1/2-inch trommel undersize is sampled by a Vezin sampler which takes an 8-percent cut, and the remainder passes by gravity into 1 of 3 storage bins. From the storage bins the ore is distributed in the mill-feed bin by a belt conveyor equipped with a hand-set wooden plow for discharging from the belt. Three rack-and-pinion gates control the flow of ore from the mill feed bin onto a steel pan feeder, which carries the ore into a screen trommel with 1/4-inch openings. The plus 1/4-inch product is crushed in a no. 3 Universal jaw crusher to about 1/4 inch. The crusher discharge and the minus 1/4-inch trommel product pass by gravity to a second belt and bucket conveyor, which discharges into two trommel screens arranged in series with 1/4- and 1/8-inch openings, respectively. At this point the ore is wet-screened. The three products from the two screens comprise the feeds for coarse concentration with jigs.

The plus 1/4-inch product is fed to a 1-cell bull jig, the plus 1/8- minus 1/4-inch product to a 2-cell jig, and the minus 1/8-inch product, after being dewatered, to a 3-cell jig. All three jigs are of the Harz type. The drop between successive screens is 1 inch in the multiple-cell jigs. Table 13 gives data on jigging.

TABLE 13. - Data on jigging

	1-cell jig	2-cell jig	3-cell jig
Length of plunger stroke, in.	3/4	1/2	1/2
Strokes per minute.....	250	250	250
Depth of bed, inches.....	5 1/2	4 1/2	4 1/2
Screen size.....	3/8-inch	6-mesh	10-mesh
Total screen area, sq. in.	495	990	1,485

Concentrates are removed from the jigs at intervals by a hand scoop. Tailings from the 1- and 2-cell jigs are dewatered and crushed to about 1/8 inch in a set of 14- by 27-inch rolls. The roll discharge is returned to the trommel screens by elevator 2 for further sizing before additional jigging.

The 3-cell jig tailings are dewatered in an Akins classifier. The dewatered sands are classified for table concentration in three revolving screens arranged in series. The overflow from the Akins classifier goes to a Dorr thickener.

The three screens are 20-, 40-, and 80-mesh, respectively. The plus 20-mesh product is treated in a Richards jig equipped with a 6-mesh screen. This jig makes 2 products, a hutch concentrate and a tailing which is reground in the ball mill. The remaining 3 screen products are treated separately on three Wilfley tables. Each of the tables makes 4 products - first- and second-grade concentrates, middlings which go to elevator 3 for regrinding, and a tailing which usually is sent to the Dorr thickener. If low-grade ore is being treated, however, the table tailings may be diverted at this point to the tailings pond without further treatment.

The table middlings and the Richards jig tailings are discharged from elevator 3 into a second Akins classifier. The classifier overflow is sent to the Dorr thickener, and the underflow is reground in a Marcy no. 3 ball mill. About 1 ton of steel balls 1 inch to 4 inches in diameter is used in the ball mill. The ball-mill discharge is returned to the first Akins classifier by elevator 4 for further screening and table treatment.

The Dorr thickener is the starting point for concentration of the slimes. About 35 pounds of lime are added to the thickener each shift to settle the colloids. The underflow is distributed to three Deister-Overstrom slime tables, each of which makes a concentrate and a tailing. The tailings are sent to a dewatering cone, whence the underflow is sent to three additional Deister-Overstrom slime tables. The tailings from the last three tables are distributed to three canvas tables arranged in parallel. The canvas tables are not primarily part of the concentrating equipment, since they are used as a barometer to determine whether or not the concentrating tables are operating satisfactorily. The tailings from the canvas plant are diverted to a pond where they are impounded with a view to future treatment. The average grade of the tailings in the pond is about 1 percent WO_3 .

After draining in pans the jig concentrates are dried in a Sutton sand drier. Because of their fineness the table concentrates are dried in pans on an ordinary cooking stove. After being dried the two classes of concentrates are sacked separately for shipment in canvas sacks. The mill balance sheet for 1931 is shown in table 14.

TABLE 14. - Mill balance sheet, Wolf Tongue concentrator, 1931

	Weight of product, pounds	Assay WO ₃ , percent	Contained WO ₃ , pounds	Total WO ₃ , percent
Mill heads.....	2,499,970	5.72	142,998	100
High-grade concentrate recovered by jigs..	156,286	53.16	83,082	58.1
Low-grade concentrate recovered by jigs...	67,844	31.41	21,307	14.9
High-grade concentrate recovered by tables	28,589	61.02	17,445	12.2
Low-grade concentrate recovered by tables.	5,924	33.79	2,002	1.4
Tailings.....	2,241,327	0.855	19,162	13.4
Total.....	2,499,970	- - -	142,998	100.0

The following tables give costs of milling 1,250 tons of ore in 1931 and the distribution of labor, power, and supplies for the same period.

TABLE 15. - Milling costs per ton of ore, Wolf Tongue concentrator

(Tons of ore treated: 1,250. Period covered: 8 months, 1931)

	Labor	Power	Supplies	Total
Crushing and grinding.....	\$0.260	- - -	\$0.168	\$0.428
Screening and conveying.....	.519	- - -	.111	.630
Jigging.....	.779	- - -	.024	.803
Tabling.....	.519	- - -	- - -	.519
Assaying and sampling.....	1/3.300	- - -	- - -	3.300
Miscellaneous.....	- - -	\$0.096	.239	.335
Total.....	5.377	.096	.542	6.015

1/ Includes custom assaying charges on many small lots of lease ore averaging less than 200 pounds each.

TABLE 16. - Milling costs per ton of ore, in units of labor, power, and supplies
Wolf Tongue concentrator

A. Labor (man-hours per ton):

Crushing and grinding.....	0.479
Screening and conveying.....	.952
Jigging.....	1.455
Tabling.....	.479
Sampling.....	.952
Total.....	4.317

Ore treated per man-shift (tons).....1.85

B. Power (kw.-hr. per ton):

Crushing.....	8.6
Grinding.....	5.4
Screening and conveying.....	4.3
Jigging.....	2.7
Tabling.....	5.7
Total.....	26.7

Percent of total cost

C. Labor.....	38.8
Power and supplies.....	10.7
Assaying.....	50.5

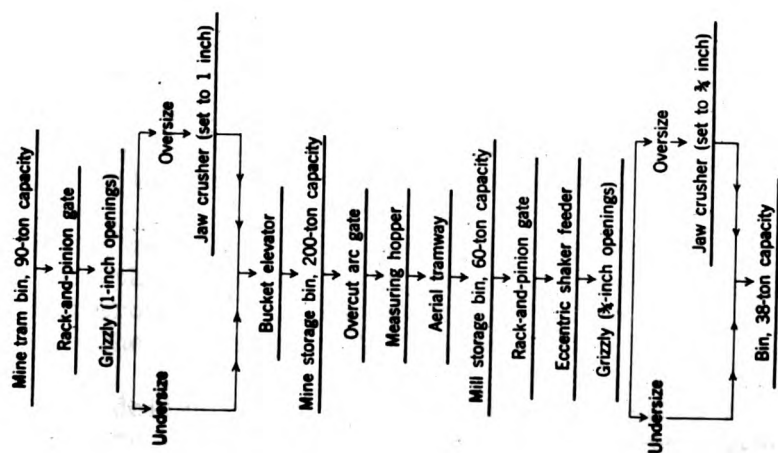


Figure 14.—Flow sheet of Silver Dyke crushing plant.

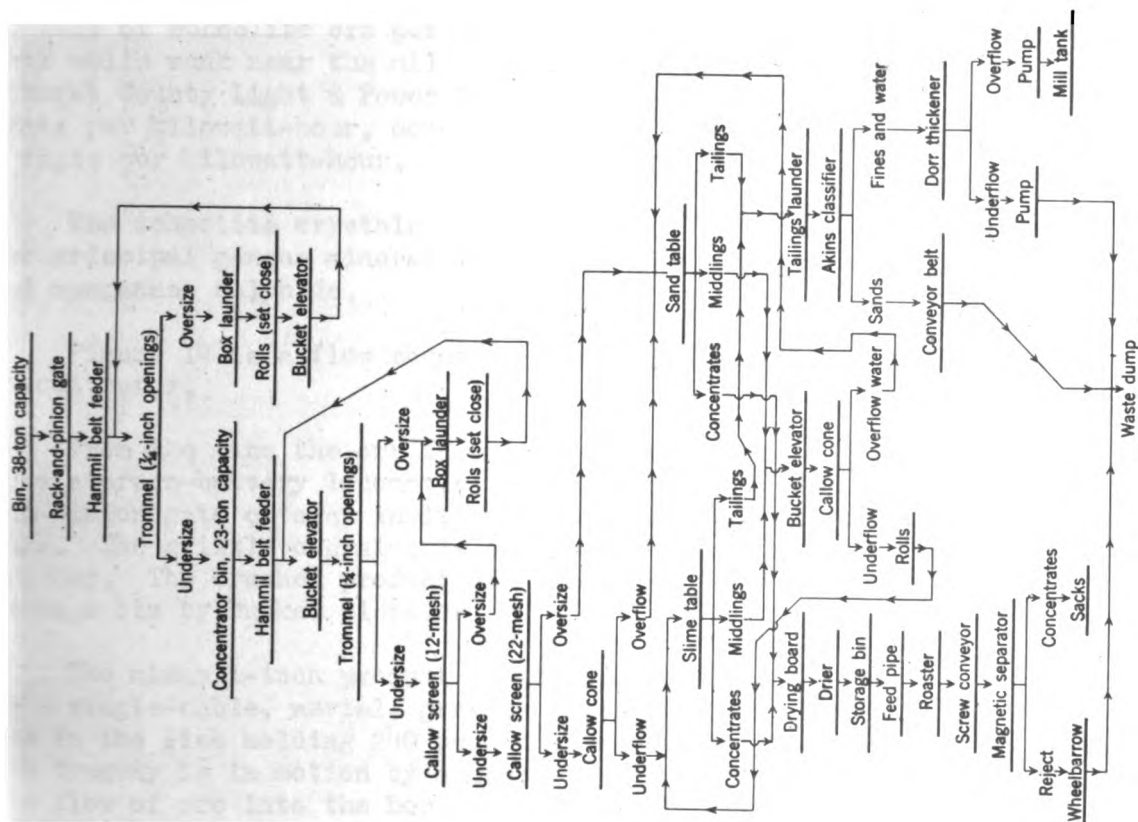


Figure 15.—Flow sheet of Silver Dyke concentrator.

Silver Dyke Concentrator

The Silver Dyke concentrator near Mina, Nev., has a capacity of about 50 tons of scheelite ore per day.¹⁵ Water for milling is obtained from several wells sunk near the mill site. Electric power is purchased from the Mineral County Light & Power Co., under a schedule ranging from 2 1/2 to 4 cents per kilowatt-hour, depending on the demand. The average cost is about 3 cents per kilowatt-hour.

The scheelite crystals range from 1/16 inch to 2 inches in diameter. The principal gangue mineral is quartz, with subordinate quantities of pyrite and manganese sulphide.

Figure 14 is a flow sheet of crushing operations and figure 15 of the concentrator.

From the mine the ore is delivered to the tram bin in 1-ton cars hauled by a storage-battery locomotive. From the bin the ore is fed through a rack-and-pinion gate onto an inclined grizzly with 1-inch openings between the bars. The grizzly oversize is crushed to 1-inch in a 16- by 24-inch Blake crusher. The crushed product and the grizzly undersize are conveyed to a storage bin by bucket elevator.

The minus 1-inch product is transported from the storage bin to the mill by a single-cable, aerial, gravity tram 1,420 feet long. There are 40 buckets in the line holding 240 pounds of ore each. The buckets are loaded while the tramway is in motion by a measuring hopper traveling on an inclined plane. The flow of ore into the hopper is controlled by a hand-operated rack-and-pinion gate. The buckets are dumped automatically at the mill bin by a tripper which releases the hinged bottoms.

From the mill bin the ore is delivered through a rack-and-pinion gate to a grizzly by an eccentric shaker feeder. Grizzly openings are 3/4 inch. The grizzly oversize is crushed to 3/4 inch in a 14- by 18-inch Blake crusher. The crushed product and the grizzly undersize pass to the mill storage bin. A Hammil belt feeder delivers the ore to a trommel having 7/16-inch holes, which operates in closed circuit with a set of 14- by 30-inch Allis-Chalmers rolls. The trommel oversize is delivered to the rolls by gravity, and the crushed product is returned to the trommel by bucket elevator.

The undersize from the 7/16-inch trommel passes to a surge bin, whence a Hammil belt feeder delivers the material to an elevator which discharges into a second trommel. This trommel is in closed circuit with a set of 14- by 30-inch secondary rolls. The secondary rolls operate wet.

The trommel undersize is delivered to a 12-mesh simplex Callow screen operating in closed circuit with the secondary rolls. The 12-mesh screen undersize passes to a 22-mesh screen, and the products from this screen

¹⁵ Vanderburg, W.O., Methods and Costs of Concentrating Scheelite Ore at the Silver Dyke Mill, Mineral County, Nevada: Information Circular 6604, Bureau of Mines, 1932, 12 pp.

comprise the feeds for sand and slime tables.

The plus 22-mesh sands and the thickened minus 22-mesh fines are each treated on an Overstrom table; each table produces concentrates, middlings, and waste tailings. The middlings from both tables are dewatered in a 4-foot Callow cone; the cone-discharge sands are crushed further by a set of 18- by 4-inch Sturtevant rolls, and the crushed product is returned to the table that treats the underize from the 22-mesh screen. The table concentrates are collected in buckets which are emptied on a drainboard. A launder carries the tailings to the dewatering plant.

The table concentrates are shoveled from the drainboard into a Channon foundry sand drier, and after drying they are dropped into a concrete storage bin. From the storage bin they are fed through a 2-inch pipe with plug-valve control to the roasting furnace. Figure 16 shows how the roaster is constructed. The concentrate as fed to the roaster contains about 19 percent pyrite, which is rendered magnetic in the presence of air by roasting for about 5 minutes. The calcines discharge into the trough of a screw conveyor which carries them to a shaker feeder at the head of a Ding's magnetic separator. The primary purpose of the conveyor is to allow the concentrates to cool before magnetic separation. The magnetic material is rejected as waste, and the cleaned concentrates are sacked for shipment.

At the dewatering plant the tailings go to an Akins classifier. The classifier sands are sent to the waste dump by a conveyor belt, and the overflow passes to a Dorr thickener for reclaiming the water. About 75 percent of the mill water is reclaimed. The thickener overflow is pumped to the mill supply tank by a 1 1/2-inch centrifugal pump; the underflow is sent to the waste dump by a diaphragm pump.

Tables 17 and 18 summarize costs and the distribution of labor, power, and supplies for a 2-month period in 1931.

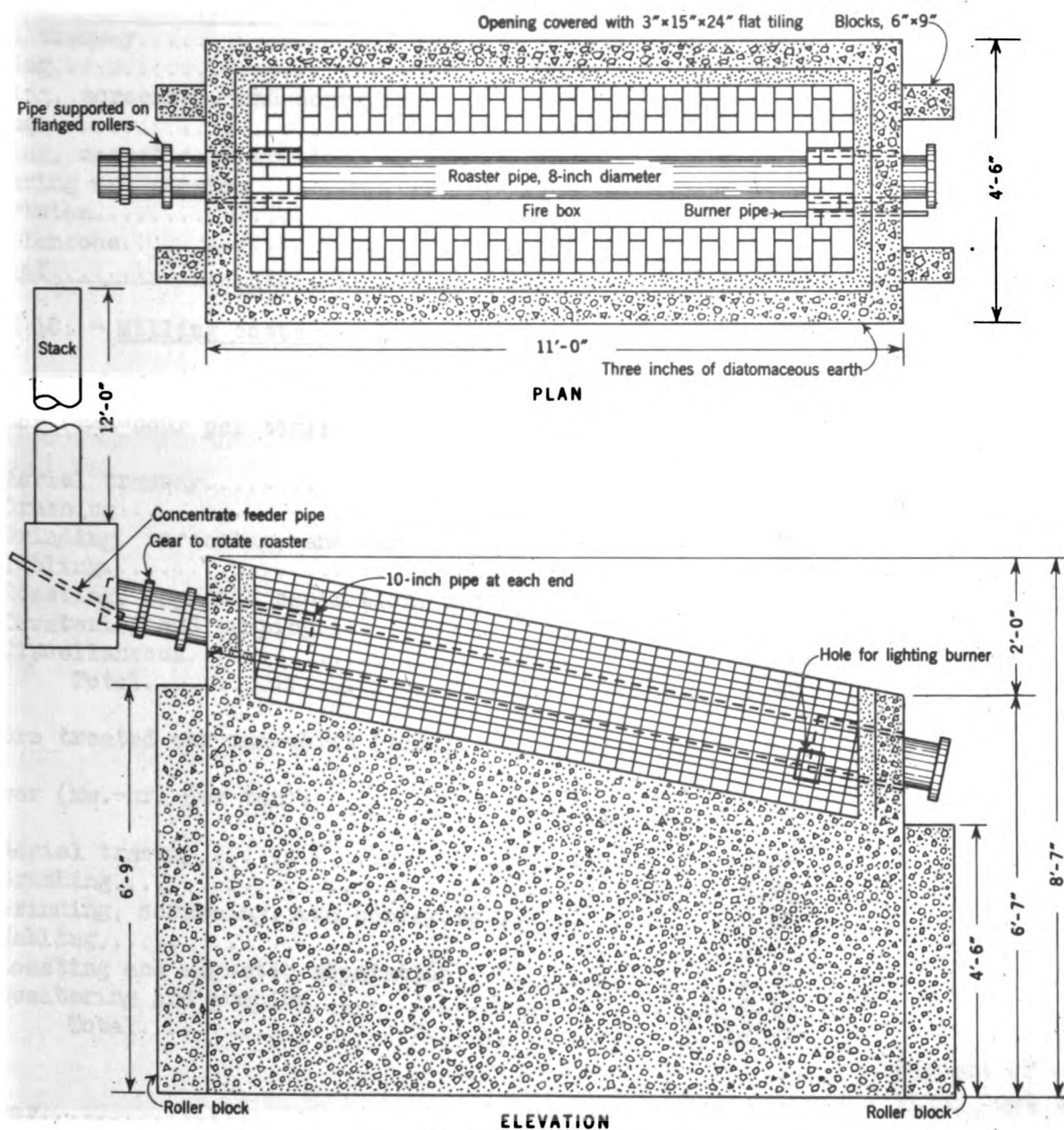


Figure 16.—Sketch of Silver Dyke roasting furnace.

TABLE 17. - Milling costs per ton of ore, Silver Dyke concentrator

(Tons of ore treated: 1,140. Period covered: 2 months, 1931)

	Labor	Power	Supplies	Total
Aerial tramway.....	\$0.104	\$0.135	- - -	\$0.239
Crushing.....	.104	.271	- - -	.375
Grinding, screening, and conveying....	.539	.432	\$0.276	1.037
Tabling.....	.208	.032	- - -	.290
Roasting, magnetic separation, sacking	.104	.033	.162	.299
Dewatering and pumping.....	.130	.135	- - -	.265
Supervision.....	.066	- - -	- - -	.066
Miscellaneous.....	.052	- - -	.026	.078
Total.....	1.107	1.078	.464	2.649

TABLE 18. - Milling costs per ton of ore, in units of labor, power, and supplies, Silver Dyke concentrator

A. Labor (man-hour per ton):

Aerial tramway.....	0.151
Crushing.....	.168
Grinding, screening, and conveying.....	.540
Tabling.....	.334
Roasting, magnetic separation, sacking.....	.184
Dewatering and pumping.....	.209
Miscellaneous.....	.084
Total.....	1.670

Ore treated per man-shift (tons).....4.78

B. Power (kw.-hr. per ton):

Aerial tramway.....	4.45
Crushing.....	8.91
Grinding, screening, and conveying.....	13.83
Tabling.....	2.66
Roasting and magnetic separation.....	1.08
Dewatering and pumping.....	4.44
Total.....	35.37

Percent of
total cost

C. Labor.....	41.8
Power and supplies.....	58.2

Mill City Concentrator

The Mill City concentrator of the Nevada-Massachusetts Co., Inc., is 8 miles northwest of Mill City, a station on the main line of the Southern Pacific Railroad.¹⁶ The concentrator has a daily capacity of 200 tons of scheelite ore containing an average of 1 percent scheelite. Power for milling is purchased from the Sierra Pacific Power Co. at an average cost of 2 cents per kilowatt-hour.

The scheelite occurs unevenly disseminated as small grains in a gangue composed principally of garnet, epidote, quartz, calcite, and pyrite. Some of the gangue minerals are more tenacious than the scheelite and in grinding tend to pulverize the scheelite, which is brittle. The aim in crushing the ore is to liberate the scheelite with a minimum of sliming; this is accomplished by stage reduction with rolls. The concentrator flow sheet is shown in figure 17.

The ore from the mines is delivered to the upper end of the mill in cars hauled by gasoline locomotive, and it is dumped into two parallel bins having a combined capacity of 500 tons. Eight rack-and-pinion gates control the flow of ore onto a 24-inch, variable-speed, steel apron conveyor which discharges on a grizzly with 1 1/2-inch openings. The oversize is crushed in a 9- by 16-inch Blake crusher. The crusher discharge and the grizzly undersize drop on a conveyor belt and thence into a belt-and-bucket elevator which feeds a trommel with 9/16-inch openings. The undersize from the trommel falls onto a belt conveyor equipped with a hand-propelled tripper and is discharged through the tripper into either of two bins, each holding 200 tons. The trommel oversize passes by gravity to a pair of 36- by 15-inch Garfield rolls set to 1/2 inch and operating in closed circuit with the trommel.

At the bottom of each bin is a Hammil feeder which discharges the ore onto a belt conveyor. The belt conveyor discharges into two belt-and-bucket elevators. The elevators feed into 1 of 2 trommels equipped with no. 23 Ton-Cap screens with 1/8- by 1/4-inch openings. The second trommel is used in case of a break-down or when making repairs. The trommel oversize goes to a set of 36- by 16-inch Gates rolls operating in closed circuit with the foregoing elevators and trommels. The set of these rolls is adjusted to the amount of circulating load; if the circulating load is increasing the rolls are set a little closer. The trommel undersize passes onto 14-mesh, Callow, duplex, traveling-belt, phosphor-bronze screen. The screen oversize is crushed in two sets of 36- by 16-inch rolls in closed circuit with the screen. The screen undersize flows to a single-spigot hydraulic classifier. The spigot delivers the coarse sands to 2 Wilfley and 4 Deister-Overstrom tables, and the overflow is sent to a second hydraulic classifier. The spigot from this classifier delivers fine sands to 3 Deister-Overstrom and 2 Wilfley tables, and the overflow goes to three 8-foot Callow cones. The Callow-cone goosenecks discharge into a simple nonhydraulic classifier. The coarse and heavier part

¹⁶ Heizer, Ott F., Methods & Costs of Concentrating Tungsten Ore at the Nevada Massachusetts Mill, Mill City, Nevada: Information Circ. 6230, Bureau of Mines, 1930, 13 pp.

of the slimes pass through a spigot to two Wilfley tables; the finer and lighter part of the slimes is pumped by a 2-inch centrifugal pump to an 8-foot Callow cone. The gooseneck product from this cone is treated on a Deister table, and the overflow from the Callow cones goes to the dewatering plant.

Each of these tables produces a bulk concentrate, a middling, and a tailing. Table middlings flow to a single Deister-Overstrom table for additional treatment. The broad middling cut from this table is pumped by a 2-inch centrifugal pump to a 4-foot Callow cone. The spigot from the 4-foot cone discharges into the fine rolls grinding the Callow-screen oversize, and the overflow joins the feed in the wet elevators.

The bulk table concentrate flows to a 2-cell Kraut flotation machine which eliminates the finer sulphide minerals, principally pyrite and molybdenite. The flotation tailing, consisting of scheelite, garnet, and the coarser pyrite, flows to a belt and bucket elevator discharging into a small Akins classifier for dewatering. The classifier overflow joins the table middling and is returned to the mill circuit.

After dewatering the bulk concentrate drops into the trough of a screw conveyor, which delivers it into the upper end of a home-made, oil-fired rotary drier. The dried concentrate drops into the boot of a belt-and-bucket elevator, which discharges into the hopper of the first magnetic separator. The concentrate is fed evenly onto the belt of a 6-pole Wetherill magnetic separator. Tramp iron, garnet, and a small amount of fine pyrite reduced in the drying process are removed by this magnetic separator. The garnet reject is carried in wheelbarrows to a separate dump. The concentrate is conveyed by a belt-and-bucket elevator and a belt conveyor into a rotary, oil-fired roaster which renders the pyrite magnetic. Roasting the garnet with the pyrite is not feasible because the garnet would agglomerate, thereby preventing magnetic separation.

The hot calcined concentrate is discharged into a steel screw conveyor where it is sprayed with a small quantity of water. The concentrate cooled in transit to the second magnetic separator, of the same size as the first, is cleaned further by removal of the magnetic iron and garnet. The rejects from this separator contain some scheelite and are returned to the mill circuit. The iron-pyrite reject is rendered porous by roasting, thus lowering its specific gravity and making it amenable to separation from scheelite by table concentration when returned to the mill circuit.

The concentrate from the second Wetherill machine goes to a belt-and-bucket elevator and thence into a steel screw conveyor, which delivers it to an externally fired roaster. The final dead roast removes virtually all remaining sulphur. After the final roast the concentrate discharges into a screw conveyor which in turn discharges into a belt-and-bucket elevator. This elevator discharges into a Dings magnetic separator, type M-2, which removes a small amount of magnetic material. The finished concentrate is elevated into a hopper by a belt-and-bucket elevator. At the bottom of the hopper is a 1 1/2-inch pipe through which the concentrate is run into double sacks of canvas and burlap.

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The dewatering plant is in a separate building. The mill tailings flow to the Akins classifier in the dewatering plant and are divided into sand and slime products in the ratio of 75 and 25. The slime flows to a Dorr thickener, which produces a clear overflow and a spigot product averaging 3.7:1 density. The slime product of the thickener is sent by gravity to the tailings pond with the classifier sands. The average pulp density of the sand-slime product is 1.15:1.

The thickener overflow is pumped to a 50,000-gallon tank above the mill, where it joins the water pumped from the mines and such additional water as may be required from the well in the valley east of the property.

Including the superintendent and repairmen, 12 men are required to operate the mill on a 3-shift basis.

Tables 19 and 20 give the direct costs of milling 40,924 tons of ore in 1928. Milling costs in 1934 were about the same as those shown in the tables.

TABLE 19. - Milling costs per ton of ore, Mill City concentrator

(Tons of ore treated: 40,924. Period covered: 1923)

	Labor	Power	Supplies	Total
Breaking and crushing to 9/16 inch.....	\$0.069	\$0.080	\$0.013	\$0.162
Crushing, screening, and conveying.....	.117	.127	.062	.306
Tabling.....	.123	.033	.004	.160
Magnetic separation and roasting.....	.098	.046	.082	.226
Dewatering and pumping.....	.053	.014	.001	.068
Superintendence.....	.066	- - -	- - -	.066
Miscellaneous.....	.047	- - -	.211	.258
Total.....	.573	.500	.373	1.246

TABLE 20. - Milling costs per ton of ore in units of labor, power, and supplies, Mill City concentrator

A. Labor (man-hour per ton):

Breaking and crushing to 9/16 inch.....	0.105
Crushing, screening, and conveying.....	.180
Tabling.....	.190
Magnetic separation and roasting.....	.154
Superintendence.....	.072
Dewatering.....	.078
Miscellaneous.....	.081
Total.....	.860

Ore treated per man-shift (tons).....9.4

B. Power (kw.-hr. per ton):

Breaking and crushing to 9/16 inch.....	4.00
Crushing, and screening.....	5.00
Tabling.....	1.66
Magnetic separation and roasting.....	2.31
Dewatering.....	.69
Conveying and elevating.....	1.34
Total.....	15.00

	Percent of total cost
C. Labor.....	46.8
Power and supplies.....	53.2

OTHER METHODS OF CONCENTRATION

Centrifugal Concentration

Experiments have been made to save scheelite from slime tailings by centrifuges. In the laboratory experiments described by Doerner¹⁷ the slime tailings constituted 20 percent of the total mill tailings and averaged about 0.75 percent WO_3 . A recovery of 60 percent on the slime portion of the tailings was possible when a concentrate containing 10 percent WO_3 was produced and the ratio of concentration was 24:1. A recovery of nearly 50 percent was possible when a concentrate containing 20 percent WO_3 was produced and the ratio of concentration was 30:1. Higher recoveries may be expected with a lower-grade rougher product recleaned in a second machine.

Although centrifuges are used widely in separating immiscible liquids or in separating solids from liquids, the application of centrifugal concentration to solid particles having different densities as yet has not been successful on a commercial scale.

Pneumatic Concentration

Several mines have concentrated tungsten ores by pneumatic methods. It is interesting to note that perhaps the first tungsten concentrates produced in the United States were concentrated by such a process at the Trumbull mine near Long Hill, Conn. The tungsten minerals, principally scheelite, occurred at a contact of limestone and gneiss; the associated minerals were chiefly garnet, quartz, hornblende, epidote, and pyrite. The ore, which contained about 5 percent scheelite, was crushed in stages with rolls. After classification by screening to 40-, 60-, and 90-mesh the products were fed to six Hooper pneumatic machines, which delivered a clean product, leaving little scheelite in the tailings. Hobbs¹⁸ states that each machine could handle 10 tons of pulp per day, and the final product contained 65 percent WO_3 .

Pneumatic concentration of tungsten ores was applied as recently as 1924, when a small test plant was erected on the Summerfield property, near Mina, Nev.¹⁹ Here the ore contains scheelite disseminated through beds of garnetized limestone. At this plant the ore after being crushed was ground to minus 10-mesh in a ball mill. The ball-mill discharge was fed to a Stebbins dry table having a capacity of 1 ton of ore per hour. The table yielded three products; a scheelite concentrate containing 15 percent WO_3 , a middling, and a waste tailing. The middling was returned to the ball mill, and the concentrate was cleaned on a smaller Stebbins table which produced a concentrate containing 60 percent WO_3 .

¹⁷ Doerner, H.A., Centrifugal Concentration. Its Theory, Mechanical Development, and Experimental Results: Tech. Paper 457, Bureau of Mines, 1929, 39 pp.

¹⁸ Hobbs, W.H., The Old Tungsten Mine at Trumbull, Conn.: U.S. Geol. Survey, 22d Ann. Rept., 1901, pp. 7-22.

¹⁹ Osborn, W.X., Tabling Tungsten Ores without Water: Eng. and Min. Jour., vol. 122, 1927, pp. 287-289.

About 500 tons of ore were treated in the test mill; the average grade of the ore was 1.1 percent WO_3 , which did not permit profitable operation on a small scale, and operations were suspended. The dust problem proved troublesome.

Several objections can be made to the concentration of tungsten ore by pneumatic methods. With most of the machines designed for this purpose the principal difficulty is adjustment, as change in the velocity of the air changes the product. Induced eddy currents make operation of the machines variable. The extremely fine particles of tungsten minerals are not heavy enough to drop out of the air current, and the recovery is therefore lower than that with wet gravity-concentration methods.

Flotation

Flotation, either as the principal method or as an adjunct to wet gravity-concentration methods, offers the most productive field for research in the concentration of tungsten ores. Experiments by Coghill and Clemmer²⁰ indicate that scheelite and ferberite ores containing quartz as the principal gangue mineral are amenable to flotation. Experimental work on the flotation of several tungsten ores by these investigators shows that recoveries and grades of concentrates compare favorably with those obtained by wet gravity-concentration methods.

Experimental work at the Reno Experiment Station of the U. S. Bureau of Mines, on the flotation of slimed scheelite in the mill tailings of contact-metamorphic-type ores treated by wet gravity methods, shows that although a good recovery can be made the resulting concentrate is low in grade. Additional experimental work is necessary to raise this low-grade concentrate to a commercial grade.

METHODS OF DETERMINING WO_3 CONTENT IN TUNGSTEN ORES

In the mining and milling of tungsten ores in the United States the WO_3 content is determined either by panning or specific-gravity methods. Chemical methods employed in custom assaying are not used owing to the difficulty of obtaining accurate results, the time required, and the cost. In experienced hands the pan is a reliable method because the tungsten minerals pan almost as readily as gold.

Specific-gravity methods are efficient when the gangue minerals are of low specific gravity compared with the tungsten mineral and the ore is high in grade. The method ordinarily is employed in determining the tungstic acid content of concentrates or hand-sorted ores. In the Atolia tungsten district it is accurate within 2 percent, as the gangue minerals have a fairly constant specific gravity.

²⁰ Coghill, Will H. and Clemmer, J.B., Soap Flotation of the Non-sulphides: Am. Inst. Min. and Met. Eng., Tech. Paper 445, 1932, pp. 13-16.

A 500-cm flask graduated in 5-cm units and a scale which weighs in grams and has a capacity of at least 200 grams are required. A representative sample of concentrates is ground to 10-mesh. Water is poured into the graduated flask until the 400-cm mark is reached, then a sample of concentrates weighing 200 grams is poured into the flask. The volume of water and solids in the flask is then read, and the tungstic acid content is calculated as follows:

Let T = combined volume of water and solids in cubic centimeters. Then
 $T - 400$ = volume of water displaced by 200 grams of concentrates, and
 $\frac{200}{T - 400}$ = specific gravity of the concentrates.

The percentage of tungstic acid in the concentrates is determined from table 21. The data in the table are based on the assumption that all gangue material has the specific gravity of quartz.

The Wolf Tongue Mining Co., Nederland, Colo., employs a specific-gravity method similar to the Atolia method, which is devised to control mill operations and to enable estimating the value of purchased ores so that partial settlement could be made on such ores the same day they were received instead of the lessee having to wait 3 or 4 days for settlement. The company has never purchased ores on specific-gravity determinations alone; final settlement on all custom ores has been made on custom assays. The specific gravities of ferberite ores are shown in table 22. The figures in this table are based on the assumption that the gangue minerals have fairly uniform specific gravities. For a specific gravity between any two consecutive specific gravities in the table the equivalent percentage of WO_3 can be interpolated.

At the Round Valley tungsten concentrator a very simple method is employed to keep the concentrate up to grade. A druggist's scale and a counterweight, which is used for comparison against the sample tested, are required. The counterweight is so made that a level blasting-cap box of concentrate, carrying 60 percent WO_3 (dried and well-settled) will just balance the counterweight. The cap box of concentrate to be tested (likewise dried and well-settled) is placed on the scale; if it balances the counterweight the sample is just up to grade and contains 60 percent of WO_3 , but if it lifts the counterweight the concentrate is a little above grade. If the sample is a little low in grade the concentrating tables are adjusted to make a cleaner product. The millmen become expert in this method of sampling and can estimate the grade of the concentrate closely by comparing samples with the counterweight.

Ultraviolet rays afford a simple and rapid means for determining scheelite ores and mill products. As a general rule, scheelite fluoresces bright light blue when activated by ultraviolet rays emitted by a high-tension disruptive spark between iron terminals. The ultraviolet ray is used at the tungsten mines of the Nevada Massachusetts Co., Inc. for qualitative as well as approximate quantitative determination of scheelite in ores and mill products. A description of the ultraviolet apparatus suitable for this work has been given by Andrews.²¹

²¹ Andrews, W.S., Apparatus for Producing Ultraviolet Radiation: Gen. Elec. Rev., vol. 9, 1916, pp. 317-319.

TABLE 21. - Tungstic acid content of Atolia scheelite ores, as indicated by specific gravity

Specific gravity	WO ₃ , percent	Specific gravity	WO ₃ , percent	Specific gravity	WO ₃ , percent	Specific gravity	WO ₃ , percent
3.00	15.40	4.77	61.80	5.19	68.00	5.60	73.20
3.10	18.40	4.78	62.00	5.20	68.10	5.61	73.30
3.20	21.20	4.79	62.00	5.21	68.20	5.62	73.40
3.30	24.50	4.80	62.10	5.22	68.40	5.63	73.50
3.40	28.00	4.81	62.20	5.23	68.50	5.64	73.60
3.50	31.20	4.82	62.40	5.24	68.60	5.65	73.70
3.60	34.00	4.83	62.80	5.25	68.70	5.66	73.80
3.70	37.00	4.84	63.00	5.26	69.00	5.67	73.90
3.80	39.60	4.85	63.20	5.27	69.10	5.68	74.00
3.90	42.20	4.86	63.30	5.28	69.20	5.69	74.10
4.00	44.80	4.87	63.40	5.29	69.40	5.70	74.20
4.10	47.30	4.88	63.60	5.30	69.60	5.71	74.32
4.20	49.60	4.89	63.70	5.31	69.70	5.72	74.44
4.30	51.60	4.90	63.80	5.32	69.80	5.73	74.56
4.40	54.00	4.91	64.10	5.33	70.00	5.74	74.68
4.50	56.20	4.92	64.20	5.34	70.10	5.75	74.80
4.51	56.40	4.93	64.30	5.35	70.20	5.76	74.92
4.52	56.70	4.94	64.40	5.36	70.40	5.77	75.04
4.53	57.00	4.95	64.50	5.37	70.60	5.78	75.16
4.54	57.30	4.96	64.70	5.38	70.70	5.79	75.28
4.55	57.50	4.97	64.80	5.39	70.80	5.80	75.40
4.56	57.60	4.98	65.00	5.40	71.00	5.81	75.54
4.57	57.80	4.99	65.20	5.41	71.10	5.82	75.68
4.58	58.00	5.00	65.40	5.42	71.20	5.83	75.82
4.59	58.20	5.01	65.50	5.43	71.30	5.84	75.96
4.60	58.40	5.02	65.60	5.44	71.40	5.85	76.10
4.61	58.60	5.03	65.80	5.45	71.50	5.86	76.24
4.62	59.00	5.04	66.00	5.46	71.60	5.87	76.38
4.63	59.20	5.05	66.10	5.47	71.70	5.88	76.52
4.64	59.30	5.06	66.30	5.48	71.80	5.89	76.66
4.65	59.50	5.07	66.40	5.49	71.90	5.90	76.80
4.66	59.80	5.08	66.60	5.50	72.00	5.91	76.94
4.67	60.00	5.09	66.80	5.51	72.10	5.92	77.08
4.68	60.10	5.10	66.90	5.52	72.24	5.93	77.22
4.69	60.20	5.11	67.00	5.53	72.36	5.94	77.36
4.70	60.40	5.12	67.10	5.54	72.48	5.95	77.50
4.71	60.60	5.13	67.20	5.55	72.60	5.96	77.64
4.72	60.70	5.14	67.30	5.56	72.72	5.97	77.78
4.73	61.00	5.15	67.60	5.57	72.84	5.98	77.92
4.74	61.20	5.16	67.70	5.58	72.96	5.99	78.06
4.75	61.40	5.17	67.80	5.59	73.08	6.00	78.20
4.76	61.60	5.18	67.90	----	-----	-----	-----

TABLE 22. - Tungstic acid content of Boulder district ferberite ores as indicated by specific gravity

Specific gravity for given percentages of WO ₃ in ores	WO ₃ in ore, percent	Specific gravity for given percentages of WO ₃ in ores.	WO ₃ in ore, percent	Specific gravity for given percentages of WO ₃ in ores	WO ₃ in ore, percent
2.724	1	3.362	23	4.383	45
2.748	2	3.395	24	4.448	46
2.772	3	3.432	25	4.513	47
2.796	4	3.471	26	4.578	48
2.821	5	3.510	27	4.643	49
2.847	6	3.549	28	4.709	50
2.873	7	3.588	29	4.783	51
2.899	8	3.629	30	4.858	52
2.925	9	3.673	31	4.933	53
2.951	10	3.717	32	5.008	54
2.979	11	3.761	33	5.088	55
3.007	12	3.805	34	5.177	56
3.036	13	3.850	35	5.266	57
3.063	14	3.899	36	5.355	58
3.096	15	3.943	37	5.444	59
3.129	16	3.997	38	5.553	60
3.162	17	4.046	39	5.639	61
3.195	18	4.094	40	5.745	62
3.228	19	4.155	41	5.851	63
3.263	20	4.211	42	5.957	64
3.296	21	4.267	43	6.060	65
3.329	22	4.323	44	---	--

CONSERVATION OF TUNGSTEN RESOURCES

Although present resources of tungsten ores are large they are by no means inexhaustible, and the supply must serve both the needs of the present and the increasing needs of the future. The waste in mining tungsten ores is much less than that in mining some of the other ores because tungsten deposits generally have strong walls which require minimum support. No pillars of low-grade ore are left behind, as in other types of mining.

In milling tungsten ores some waste, such as the slime loss in the tailings, is unavoidable under present economic conditions. Although these tailings now have little or no present value they should be stored for future use, because as metallurgical science advances they undoubtedly will become valuable.

Some companies impound mill tailings in compliance with the laws against stream pollution; many tons of low-grade material are thus available for future treatment. At the same time little discretion is exercised in segregating the tailings according to tungsten content, with the result that future treatment is more difficult. The slime portion of the tailings should be segregated from the sand tailings when practicable; segregation when the ore is milled involves little effort and expense and would result in greater benefit for the future. In general, tailings conserved for future treatment should not be stored over wide areas nor be so placed that they are covered by valueless material later.

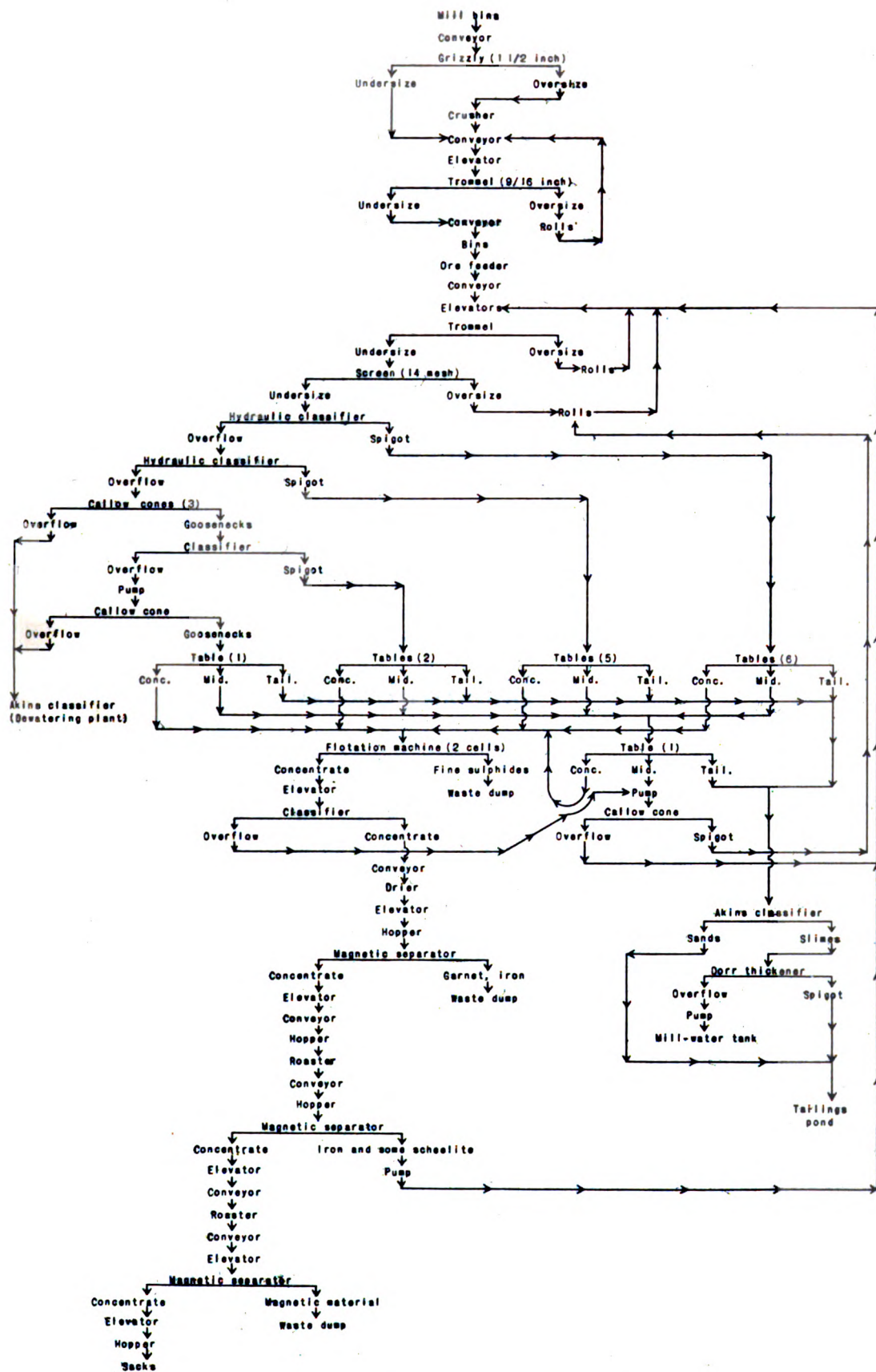


Figure 17.- Flow sheet of Mill City concentrator.